# LONGWALL DEMONSTRATION IN WESTERN CANADIAN COALS

## PHASE 1 - MINE PLANNING & SYSTEMS ENGINEERING

## CONTRACT #OSQ82-00107

## VOLUME 2

## APPENDICES - McINTYRE REFERENCES

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For

Canada Centre for Mineral
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# APPENDIX 2-1

# TERMS OF REFERENCE

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The terms of reference for the Longwall Demonstration Phase 1 Study were established in a proposal submitted to CANMET, dated June 9 1982. The terms of reference were detailed in the proposal in Section 3 Technical Proposal, Section 4 Scope of Work and Appendix 3 List of Deliverables, which are included here for reference:

## 3.0 TECHNICAL PROPOSAL

The essential requirement for successful longwall mining is compatibility between the mining equipment and mining conditions to be encountered. Accordingly, it is necessary to aquire and interpret the site specific data upon which the equipment specifications can be based.

Data derived from exploration activity can be analyzed and interpreted to provide information on major faults, seam continuity, seam thickness and general structure. However, drilling programs cannot define minor faults or rolls and these minor deviations could have an adverse effect on longwall operation. To minimize this possibility, the retreating longwall system can be adopted. With this system, roadways spaced at the planned face width are driven to a pre-determined boundary. The roadways are then connected to form the longwall face. In this manner the maximum amount of geologic information is obtained by "blocking out" the panels prior to longwall operation. The face equipment is then installed and the longwall retreated between the roadways.

If minor disturbances are detected during development, then plans can be modified to negotiate the disturbances when the panel retreats. In this way, equipment utilization can be maximized and high safety standards maintained.

McIntyre will provide Canmet with a detailed technical assessment of the planning and system design for a longwall installation. This assessment will be site specific to No. 9-G Mine at McIntyre's Smoky River operation and will be related to the operation and installation of a series of longwall retreat mining panels.

The scope of the work that will be undertaken related to geology, mine planning, system design and procurement is given in the following sections. A list of deliverable plans, schedules and other material is given in Appendix 3.

## 4.0 SCOPE OF WORK

### 4.1 Geotechnical and Geological Evaluation

This evaluation will be prepared by Norwest Consultants on behalf of McIntyre. Geological data from exploration programs conducted by McIntyre will be assessed and evaluated and geotechnical data will be derived, upon which equipment specifications will be based. Where necessary, tests will be conducted to acquire additional relevant geotechnical information.

The following will be provided:

1. The coal reserves contained in each seam suitable for longwall mining will be identified and total and recoverable tonnage calculated. Basic data and supporting calculations will be included.

2. Isopach maps at a scale of 1 : 2000 showing:

   a) seam thickness for #4 and #10 seams
   b) seam cover for #4 and #10 seams

3. The #4 seam contains a thin band of sheared coal. An isopach map showing the location and thickness of this zone in the seam will be prepared.

4. Based on borehole data, a fence diagram showing correlation of strata and peculiar coal seam features will be prepared.

5. The caving characteristics of the strata immediately above the seam will be examined and an assessment made of these characteristics related to longwall operations.

6. An assessment of the anticipated methane emissions from longwall operations will be made.

7. The potential for spontaneous combustion will be assessed.

8. For the mining areas selected, the proposed mining sequence will be reviewed and comparison of extraction and overall recovery ratios between longwall and conventional mining methods will be provided.

9. The effect of longwall extraction techniques on adjacent seams and surface structures will be evaluated and a prediction of the likely effects provided.

10. Based on the foregoing evaluations, an assessment of the feasibility of longwall applications in both seams will be made. Recommendations for future geological/geotechnical work will be provided.

11. Recommendations for studies that can be undertaken during the demonstration, to assist any further longwall applications, will be made.

## 4.2 Mine Layout

A rationale for the adoption of longwall retreat mining as the most appropriate system will be provided together with a general description of longwall advance, longwall retreat and proposed development systems. An evaluation of No. 9G mine's suitability to accept longwall operations will be made.

The following additional information will be provided:

1. A general site plan showing the location of the mine relative to other mining areas and coal preparation plant. This plan will also show coal transportation routes and the main power supply system.

2. A plan showing surface structures and stores facilities, etc.

3. A composite underground five year mine layout.

4. An operational action program for a five year period.

## 4.3 Mining Systems

Based upon the five year mine layout and action program, detailed information will be provided for the following activities:

### 4.3.1 Development of Longwalls

The following will be provided in detail for the development of longwall panels 1 and 2.

1. A development plan
2. An action program

3. Roof support methods and support rules
4. Construction details at critical areas such as air crossings, junctions and transfer points
5. Coal clearance facilities
6. A plan showing ventilation arrangements for main and auxilliary ventilation systems
7. A plan showing auxilliary equipment installations, i.e. main power system, fire fighting, water control
8. Development equipment specifications and performance data
9. Manpower deployment schedules

4.3.2 Ventilation

The ventilation requirements for the mine will be presented, supported by calculations and plans. The plans will show the ventilation of each development, the first longwall face and succeeding faces. Major changes to the ventilation system, with timing, will be identified.

An assessment of the ventilation requirements for multi-seam operation will be made and ventilation plans provided.

### 4.3.3 Supplies

The logistics of providing an adequate supply of materials to support development work and longwall mining activity will be examined. The requirements for each activity will be defined and a suitable supply system designed. Typical supplies consumption for development and longwall operations will be developed.

Stock levels to be maintained at the mine surface and at selected locations below ground will be determined and areas for storage of equipment below ground identified.

Essential machinery spares will be identified for each activity and suitable methods for storage and retrieval will be specified.

### 4.3.4 Transport

A suitable transportation system capable of handling men, materials and supplies for both development and longwall activity will be specified.

Particular attention will be directed toward the specification of a system to ensure rapid installation and withdrawal of longwall face equipment.

For the longwall retreat operation, a system of moving the ancilliary equipment associated with the longwall will be specified.

4.3.5 Power Supply System

The power system for the mine will be specified and drawings showing the layout for development and longwall operation will be provided.

Power distribution equipment will be specified and an installation schedule prepared.

4.3.6 Water Management

An estimate of the likely water inflows into the mine will be made and a scheme to handle this water will be prepared. The required equipment will be specified and an installation schedule prepared.

### 4.3.7 Safety and Fire Control

A suitable system for fire precautions, fire fighting and rescue will be developed and specified to fully comply with Provincial requirements; this will include equipment specifications. A plan showing the required layout will be provided.

### 4.3.8 Maintenance

A maintenance plan and equipment maintenance schedule will be developed.

The level of skill among existing staff will be assessed and the skilled manpower requirements to achieve a satisfactory level of maintenance will be identified. Training needs will be identified by skill.

Basic maintenance schedules and minimum spares levels for each major item of equipment will be identified.

## 4.4 Longwall Specification and Bid Assessment

The specification of longwall equipment suitable both for #4 and #10 seams will be developed in sufficient detail to allow bidding action to be initiated.

Based on the specification, an enquiry document will be prepared and selected equipment suppliers will be invited to tender for the supply of the equipment.

The tenders received will be evaluated taking due account of technical compliance with the specification, supplier's performance record and financial stability, delivery schedules, proposed financing arrangements, etc. An assessment of the tenders received will be made and a supplier recommended. A summary of the tenders will be provided.

## 4.5 Longwall Operations

Schedules for the operation of the initial two panels (Panels 1 and 2) will be provided in detail.

### 4.5.1 Panels 1 and 2

The following information will be provided:

1. Plans showing phasing of extraction

2. Production schedule
3. Installation schedule
4. Withdrawal and reinstallation schedule
5. Materials consumption schedule
6. Manpower and shift schedule
7. Projected performance levels calculated using appropriate parameters for equipment performance, seam thickness, shift patterns, etc.
8. The skill levels of existing work force will be evaluated and training needs identified by task.
9. Recommendations to satisfy training needs will be made and an estimate of the cost of suitable training programs provided.

4.5.2 Multi-Seam

A schedule showing possible multi-seam application of the longwall equipment will be prepared.

## 4.6 Capital and Operating Costs and Financing

Based upon the evaluation of tenders and selected tender prices, the following data will be provided:

1. Initial capital cost schedules
2. Replacement capital cost schedules

3. At production levels derived in Section 3.5.7, operating cost statements will be developed.
4. At varying production levels, cash flow statements will be developed and presented.
5. Financing options for Phase 2 will be reviewed and recommendations made.

## 4.7 Project Organization

To ensure that Phase 2 of the project (the demonstration) provides maximum benefit to the mining industry, it must be closely controlled and coordinated. A suitable organization structure will be proposed and the required skill and experience level of the members will be specified. Due consideration will be given in developing this organization to the need to ensure close liaison with interested mining companies and government agencies.

In addition, a method for retrieval and storage of suitable data derived during the project life will be recommended, together with areas of additional research that may be usefully undertaken.

## 4.8 Deliverables

A list of specific plans, schedules, etc. that will be delivered with the final report is given in Appendix 3.

# APPENDIX 3

## LIST OF DELIVERABLES

### Plans and Diagrams

**Geology and Geotechnical**

- Reserves #4 Seam with Isograde Lines
- Reserves #10 Seam with Isograde Lines
- Seam Thickness Isopach #4 Seam
- Seam Thickness Isopach #10 Seam
- Seam Cover Isopach #4 Seam
- Seam Cover Isopach #10 Seam
- Caprock Isopach #4 Seam
- Sheared Coal Zone Isopach #4 Seam
- Fence Diagram Showing Coal Features
- Typical Seam Sections and Adjacent Strata #4 Seam
- Typical Seam Section and Adjacent Strata #10 Seam
- Projected Sub-Surface and Surface Convergence Mining #4 Seam

**Mining Layout**

- General Minesite Plan Showing Transportation Routes, Power System, etc.
- 9G Mine Surface Arrangement Plan
- 5 Year Mining Plan

**Mining Support Systems Development**

- Detailed Development Plan Panels 1 and 2
- Roof Support Plan
- General Arrangements of Air Bridges
- General Arrangements of Junctions
- General Arrangements of Transfer Points
- General Arrangements of Coal Clearance System
- Auxilliary Ventilation Systems
- Mining Services Plan

**Ventilation**

- Plan Showing Ventilation to each Development
- Plan Showing Ventilation to Panel 1
- Plan Showing Ventilation to Succeeding Panels
- Plan Showing Ventilation for Multi-Seam Operation

| | |
|---|---|
| Supplies System | Plan Showing Supplies Route, Location of Routine Items and Essential Spares |
| Transportation | Plan Showing Routes for Men, Materials and Supplies for Development and Longwall<br>Plans/Diagrams Showing Methods for Transportation of Longwall Equipment<br>Plan of Coal Transportation System Longwall |
| Power Supply System | Plan of High Tension Distribution System<br>Plan Showing Development Power Supply<br>Plan Showing Longwall Power Supply |
| Water Management | Plan Showing Pumping Stations and Pipeline Route with Sizes, Quantities, etc. |
| Safety Fire Control | Plan Showing Fire Fighting Line with Hydrants, Location of Extinguishers and First Aid Equipment, etc. |
| Longwall Panels 1 and 2 | Plan Showing Phased Extraction for Panels 1 and 2<br>Drawings Showing Operating Methodology<br>Drawings Showing Equipment Movement Methods<br>Drawings Showing Face End Operations |
| Project Organization | Drawing Showing Suggested Structure for Project Control |

## Schedules/Tables

| | |
|---|---|
| Geology | Reserves Supporting Calculations<br>Geotechnical Data Tables |
| Mining Layout | 5 Year Action Program |
| Support Systems Development | Development Action Program Panels 1 and 2<br>Development Equipment Schedule<br>Development Equipment Performance Table<br>Development Manpower Deployment Schedule<br>Development Cost Schedule |
| Ventilation | Supporting Calculations |
| Supplies System | Recommended Stock Levels<br>Essential Spares Requirement |
| Transportation | |
| Power Supply System | Equipment List<br>Installation Schedules |
| Water Management | Equipment List<br>Installation Schedule |
| Safety and Fire Control | Equipment List<br>Installation Schedule |
| Maintenance | Maintenance Schedules - Development Longwall<br>Maintenance Training Schedules |

Longwall

| | |
|---|---|
| Panels 1 and 2 | Equipment Specification<br>Enquiry Documents<br>Tender Documents<br>Tender Summary Table<br>Production Schedules<br>Installation Schedules<br>Withdrawal and Reinstallation Schedule<br>Materials Consumption Schedules<br>Manpower and Shift Schedules<br>Performance Schedules<br>Recommended Training Schedules<br>Training Cost Schedules |
| Multi-Seam | Multi-Seam Extraction Schedule |
| Cost | Initial Capital Cost Schedule<br>Replacement Capital Cost Schedule<br>Operating Cost Schedule<br>Cash Flow Tables |

# APPENDIX 2-2

# GEOLOGY, COAL QUALITY AND RESERVES

# APPENDIX 2-2 - GEOLOGY OF THE SMOKY RIVER COAL LEASES

## STRATIGRAPHY

The Lower Cretaceous strata containing the coal seams extends northwest-southeast from the Smoky River in the Rocky Mountain Foothills. Correlation within this area has been difficult since the sedimentary environment changes between the Pine and Athabasca Rivers.

South of the Smoky River, in the vicinity of the Athabasca River, the Cadomin Conglomerate is overlain by a thick succession of alluvial sandstones, siltstones and mudstones assigned to the Luscar Formation (Table 1). Mineable coal seams occur in the lower part of the Luscar. This formation is overlain by marine shales and sandstones of the Shaftesbury Formation.

North of the Smoky River, along the Pine River in North Eastern British Columbia, two marine successions are included in the strata equivalent to the depositional period of the Luscar Formation. These are the Gething Formation, and the Gates and Boulder Creek Members of the Commotion Formation which comprise alluvial sandstones, siltstones and mudstones with mineable coal seams in the Gething and Gates. The Moosebar Formation and the Hulcross Member area contain marine shale deposits.

The Smoky River coalfield lies between these two areas. Recent work indicates that the "Big Mudstone" underlying Seam #4 is equivalent to the Moosebar marine shales. Seams #1, #1.5 and #2 therefore should be assigned to the Gething Formation. Coal Seams #3 through #11 fall

within the Gates Member of the Commotion Formation. The marine Hulcross Member does not seem to extend into the Smoky River Area.

The stratigraphy in this area reflects a period of marine incursion during the time of coal deposition. The coal quality therefore, could differ markedly between Gething and Gates coals based on different depositional environments.

## STRUCTURE

The Smoky River coalfield lies within the Foothills section of the Rocky Mountain Fold-and-Thrust belt. The structural framework is a succession of south-westerly dipping thrust faults (reverse faults) that show a displacement of hanging wall up and to the northeast (Figure 1). The strata between the major thrust faults have been deformed into north-westerly trending folds dissected by minor thrust faults.

The folds and thrust faults are intimately related in the overall shortening of the stratigraphic pile. Since folding is mainly of the concentric style, thrust faults pass laterally into the cores of anticlines where the leading fault edge becomes capped by the fold, or faults arise out of the cores of synclines and form "back-limb" thrusts on the adjoining anticlines. While major thrust faults dip fairly steeply in the area the angle of intersection between faults and bedding is small, that is less than 30°. Minor thrusts have been observed to intersect bedding at less than 10°.

The folds are predominantly concentric, that is a great deal of the deformation is accommodated by bedding plane slip. This imparts a

strong bedding anisotropy to the rock mass, little cohesion across bedding planes, particularly in the competent interbedded sandstones and siltstones.

Overall the structures are quite persistent along the north-westerly trend. However, across the structural grain in a north-easterly direction, dips change dramatically and fault repetitions are common.

## GEOLOGY OF THE COAL BEARING SUCCESSION

Nine coal seams of potentially mineable thickness have been identified within the Smoky River coalfield. Only Seam #4 is consistently present throughout the area, all other seams are limited to different parts of the coalfield. Three coal seams (Seams #4, # 10 and #11) occur in the northern part where mining is currently taking place between the Muskeg and the Mason Thrust faults. These seams fall within the lower part of the Luscar Formation a succession of interbedded alluvial sandstones, siltstones and mudstones, (Gates equivalent, Table 1).

Large scale structures (faults and folds) form natural boundaries for individual mining areas while small scale structures complicate the mining process, particularly in underground operations. Jointing, small thrust faults, and folds occur throughout the coal bearing succession.

The one master joint set in the area is oriented near vertical in a north-easterly direction, perpendicular to the fold axis and parallel to the slip on the thrust faults. These extension joints form in zones, a few feet wide, separated by tens of feet. They are best developed in the more competent sandstone and siltstone units and are, in

many localities, absent from the mudstone/shale units. The excellent caving characteristics experienced in No. 2-11 Mine, where a massive sandstone overlies the seam, was attributed to this master joint set. Other joint sets exist, but these are confined to small areas and specific structural settings.

A few minor thrust faults have been encountered in the underground workings. Small scale folds or rolls can be found in homoclinal successions and have been encountered in No. 2R-4, No. 2R-11 and No. 9A-4 Mines. The amplitude and strike length can be quite variable. Small folds may be associated with thrust faults, capping the leading or trailing edge of the fault. This situation was experienced in No. 2R-4 Mine where an open anticline was encountered in an easterly directed entry and development showed the fold tightening up and finally being cored by a thrust fault.

**Seam #4**

Seam #4 lies about 660 ft above the Cadomin Conglomerate. The stratigraphic thickness ranges from 5 to 35 ft and overall there is a general increase in thickness from southwest to northeast of about 10 ft at No. 1 Mine, 15 ft at No. 7 Mine and 20 ft at No. 2 Mine. (Figure 1). Superimposed on this regional thickness trend are the structurally induced thickness variations. These have been documented in No. 9 Surface Mine whereas the average thickness of Seam #4 is 21 ft. However in the fold hinges, unfaulted seam sections of over 60 ft have been encountered and one faulted fold produced a seam section of over 80 ft. These excessive thicknesses must be accompanied by a relative thinning elsewhere. This usually occurs in the limbs of the folds close to the fold hinge. The minimum thickness encountered in the open

pit is 3.5 ft. Thickness variations in the underground mines have not been documented in detail since only the upper part of the seam is mined. However, in No. 2-4 Mine the stratigraphic thickness of the seam was checked in five boreholes over a strike distance of 3000 ft and the range in thickness was 22 to 26 ft.

The roof of Seam #4 changes in lithology within the Barrett-McEvoy structure. The immediate roof consists of a thin interval of interbedded carbonaceous shale, dirty coal and coal, known locally as caprock. No. 5 Mine was the only area where 2 ft of caprock was present throughout the mine. Westward, in No. 2, No. 2R and No. 9 Mines the caprock is quite thin or totally absent. Overlying the caprock in No. 5 Mine is a massive mudstone unit which is dissected by numerous curved, carbon-coated fractures. In the other mines the caprock, if present, is overlain by bedded silty mudstones or siltstones (Figures 2, 3, 4 and 5). In No. 9 Mine area the roof strata tend to be more competent siltstones and sandstones.

The floor of Seam #4 consists of siltstone or sandstone throughout the mining area.

**Seam #10**

Seam #10 lies about 200 ft stratigraphically above Seam #4 and has not been mined extensively in open pit or underground. Seam thickness varies from 10 to 13 ft of which the upper 3 ft consist of coal, shale, dirty coal and bone coal.

The roof consists of fissile shales interbedded with thin coal partings. This unit ranges in thickness from 9 to 18 ft and is overlain by thick sandstones or siltstones.

The floor of Seam #10 is a massive siltstone which seems to be quite persistent in the mining area. The seam stratigraphy in No. 2-10 Mine is shown on Figure 6.

**Seam #11**

Seam #11 lies about 300 ft above Seam #4 and 100 ft above Seam #10. Although Landes (1963) described this seam as erratic, it is quite persistent in No. 2-11 Mine along a strike length of two miles with the thickness ranging from 7 to 12 ft. Where Seam #11 has been tested in No. 9 Surface Mine the thickness has varied from 7 to 13 ft. However, in the No. 9A and No. 9G Mine areas the seam shows a high variation in thickness. Seam #11 is absent in one drill hole, 4 to 7 ft thick in a few holes and shows a fairly normal thickness of 8 to 10 ft in others. There is insufficient information at this time to establish any trends to this seam variation. The seam stratigraphy is shown in Figures 7, 8 and 9.

The roof of Seam #11 consists of a massive sandstone more than 50 ft thick near the Smoky River portals of No. 2-11 Mine. The basal section of this sandstone unit grades laterally into an interbedded succession of mudstones and siltstones. This occurs both in an easterly direction toward No. 5 Mine, and a westerly direction toward No. 2R and No. 9 Mines.

The coal seam consists of dirty coal and contains a carbonaceous shale band which 'pinches and swells' from a few inches to almost two ft.

The immediate floor of Seam #11 is a carbonaceous mudstone, soft in places, which is underlain by 10 to 20 ft of interbedded siltstones and mudstones. In No. 2-11 Mine this interbedded unit is underlain by a 2 to 4 ft coal seam which contributed significantly to the floor heave encountered in this mine.

## COAL QUALITY

Coking tests with clean coal from individual seams and blends of all three seams have demonstrated the excellent coking characteristics of Smoky River coals. In commercial coke making processes they are used as a blend constituent to improve the coke quality of medium and high volatile coals.

The properties of the clean coal vary slightly from the lower to the upper seams as the rank of the coal lowers. Reflectance reduces and volatile matter increases, raising the values of FSI and fluidity. There is also a trend to higher ash contents in the upper seam.

There are indications that the properties of clean coal from Seam #4 change in a north to south direction, but the trend in the existing mining areas is so small that it has little impact on product quality.

### Seam #4

Seam #4 is a high-quality, low-volatile bituminous coking coal with an ASTM rank of Lvb. Its ISO designation is hard coal class 3-4 with an

ISO number 333-433. The coal has a Gray-King number of G4-G5 and a Roga Index of over 45. Cleaned coal at an ash content of 7% has an FSI in the range of 7 to 9. Sulphur content is less than 0.5% and its volatile matter ranges from 18 to 19%. Giesler fluidities are less than 20. The reflectance values range from 1.44 to 1.66. Table 2 shows the range of coal quality figures for Seam #4 as determined from the numerous coal samples which have been tested as part of the exploration programs. The analyses shown in Table 2 are on a common clean coal basis.

**Seam #10**

Seam #10 is also a high-quality, low-volatile bituminous coking coal with general qualities very similar to Seam #4. The reflectance of Seam #10 is lower than Seam #4 at 1.52-1.56, demonstrating its lower rank, as would be anticipated by its higher position in the stratigraphic sequence. Consequently volatile matter levels are approximately 1% higher than Seam #4 at 19 to 20% and Giesler fluidity increases to approximately 70.

The ash content of clean coal from the total seam is slightly higher, at the same separating gravity, than Seam #4. This is due to banded shale and carbonaceous shale occuring in the upper three ft of the coal seam. Exclusion of this section of the seam results in the clean coal ash content being similar to Seam #4.

**Seam #11**

Seam #11 is the uppermost seam in the coal measures and also shows excellent coking characteristics. The rank of Seam #11 is slightly

lower than the lower seams which is shown by reflectance values ranging from 1.42 to 1.48, higher volatile matter (20%-22%), increased Giesler fluidity (up to 420) and FSI (8-9).

The presence of rock and carbonaceous shale bands reduces the yields of clean coal and also increases its ash content at the same specific gravity compared with Seam #4 and #10.

## COAL RESOURCES

A clear distinction is made between the assurance that a particular coal deposit exists and the technical feasibility of extracting that coal.

The term 'resources' refers to all coal deposits. Resources are subdivided into categories that reflect the level of confidence in their existence. The term 'reserves' is limited to that part of the resources which can be mined (Table 3).

The following resources definitions have been extracted from the Canadian Department of Energy, Mines and Resources' publication "Coal Resources and Reserves of Canada" (Report ER 79-9) Appendix B:

> Assurance of Existence
>
> The terms measured, indicated, inferred . . . denote the level of confidence with which given quantities of resources have been determined or estimated; they are defined as follows:
>
> Measured Resources are resources for which tonnages are computed from information revealed in outcrops, trenches, mine workings and boreholes. The spacing of points of observation necessary to justify confidence in the character and continuity of coal seams differs from region to region according to the

character of the deposits and the geological conditions. In general the points of observation should be separated by less than the following distances:

Cordillera*.....300 m (1000 ft)
(150 m (500 ft) in severely contorted areas)

* Cordillera region includes all British Columbia and the Foothills and Mountain regions of Alberta.

Indicated Resources are resources for which tonnages are computed partly from specific measurements and partly from reasonable geological projections. For the general coal regions in Canada, the points of observation should be separated by less than the following distances:

Cordillera.....600 m (2000 ft)
(300 m (1000 ft) in severely contorted areas)

Inferred Resources are resources for which quantity estimates are based largely on broad knowledge of the geologic character of the bed or region and for which few measurements of seam thickness are available. The estimates are based primarily on an assumed continuity of coal seams in areas remote from the points of observation used to calculate measured or indicated resources.

The concepts expressed in this document are largely those applied in this presentation. However the limits of the resource categories must reflect the geologic knowledge of the Smoky River coalfield. Pertinent to resource evaluation is the greater continuity of structures along strike as compared to the dip direction. This has been incorporated in the limits for measured, indicated and inferred resources set out below:

- Measured: Extrapolation beyond a data point should not exceed 1000 ft along-strike and 500 ft across-strike.

- Indicated: Extrapolation beyond a data point should not exceed 2000 ft along-strike, and 1000 ft across-strike.

- Inferred: Based on geological knowledge extrapolations can be made along-strike but should not exceed 1000 ft across-strike.

These limits result in elliptical areas of influence. It should be noted that for the inferred category extrapolation across-strike does not exceed that of the indicated category. These limits are maximum values. Where structures or stratigraphy are known to change more conservative distances are used.

Coal quantities in any given deposit are determined by the areal extent and thickness of a seam. This gives a volumetric measure of the resources. Since tonnages are required the specific gravity of the coal must be determined. In the Smoky River coalfield two values of specific gravity have been used, 1.38 for all underground mines and 1.35 for the surface mines. These figures are based on the density of the coal itself and are a conservative estimate for the total seam.

**Resource Compilation**

A compilation of resource estimates for the Smoky River coalfield is given in Table 4. The coal resources are listed by area and whether they are potentially mineable by surface or underground methods. The underground section is sub-divided into moderate and steep dip. Moderate dip generally refers to coal that could be potentially mined by current room and pillar methods. Those coal deposits that will require new technology are listed as steep dip. Two sources were used for this compilation: summary tables from a report by the John T. Boyd Company in 1976 and figures generated by McIntyre Mines Limited in 1979, 1980 and 1981.

The grand totals shown in Table 4 represent a very significant resource base for the Smoky River operation summarized as follows:

| | Million Tons Raw Coal |
|---|---|
| Surface | 59 |
| Underground Moderate Dip | 164 |
| Steep Dip | 635 |
| Total | 858 |

It is clear that the measured resources potentially mineable by surface methods and current underground methods form a substantial basis for the operation. In addition, the combination of developing technology and further exploration could bring about an order of magnitude increase in the reserves of the property.

The identified coal resources outside those mine areas envisaged as the source of production in the near term include:

- Mine 1: A total of 196 million tons of underground resources at steep dip.

- Mine 3: A total of 54 million tons of underground resources at steep dip.
- Mine 7: A total of 15 million tons with 10 million tons in Seam #4 and 5 million tons in Seam #3. The deposit is flat-lying and amenable to recovery by underground mining methods.

- Mine 14: A total of 119 million tons of which 20 million tons is in the surface category and 99 million tons in the underground category, of moderate or steep dip.

The Smoky River mines have more than sufficient resources to ensure a reliable long-term supply of metallurgical coal.

TABLE 1

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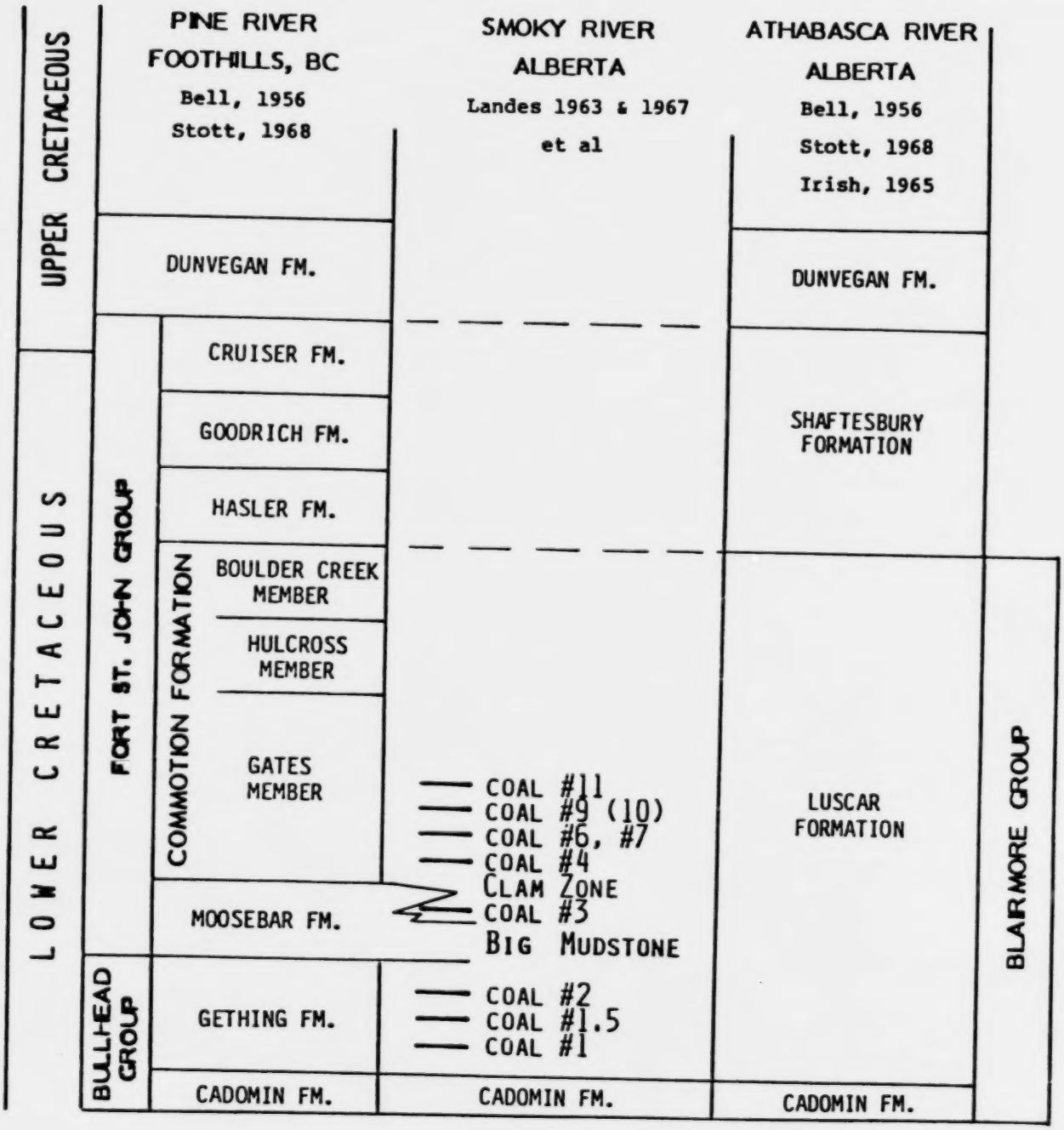
STRATIGRAPHIC CORRELATION




McINTYRE MINES

TABLE 2

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TYPICAL CLEAN COAL QUALITY - #4 SEAM

(Analysis on Dry-Basis)

| | |
|---|---|
| Ash Content % | 6.0 - 7.5 |
| Volatile Matter % | 18 - 20 |
| Fixed Carbon % | 72 - 75 |
| Sulphur | 0.3 - 0.5 |
| Btu | 14400 - 14600 |
| Ash Fusion (°F) | 2650 - 2750 |
| Hardgrove Grindabiity | 84 - 100 |
| Free Swelling Index | 7 - 9 |
| Gieseler Fluidity (DDPM) | 1 - 20 |
| Coke Stability (+1") | 54.5 - 66.0 |
| Coke Hardness (+1/2") | 55.0 - 67.5 |
| Shatter (+2"%) | 88 - 97 |
| Micum Tumbler | 73 - 83 |
| JIS Tumbler Index | 90 - 95 |

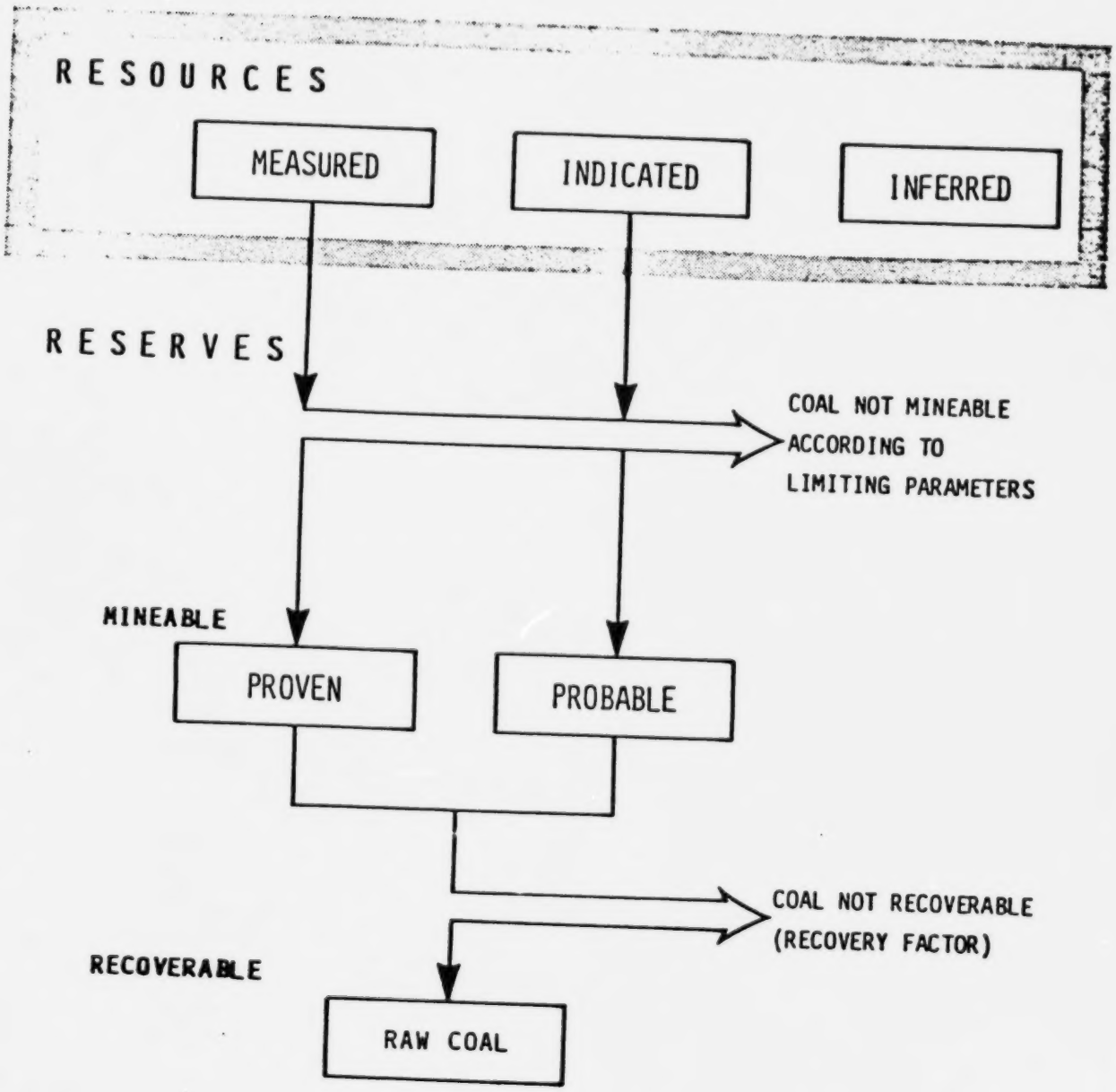
Petrographic

| | |
|---|---|
| Total Reactives | 64 - 72 |
| Total Inerts | 28 - 36 |
| Reflectance ($R_o$) | 1.44 - 1.66 |

TABLE 3

McINTYRE MINES LIMITED

## RESOURCES AND RESERVES TERMINOLOGY



# TABLE 4

## McINTYRE MINES LIMITED

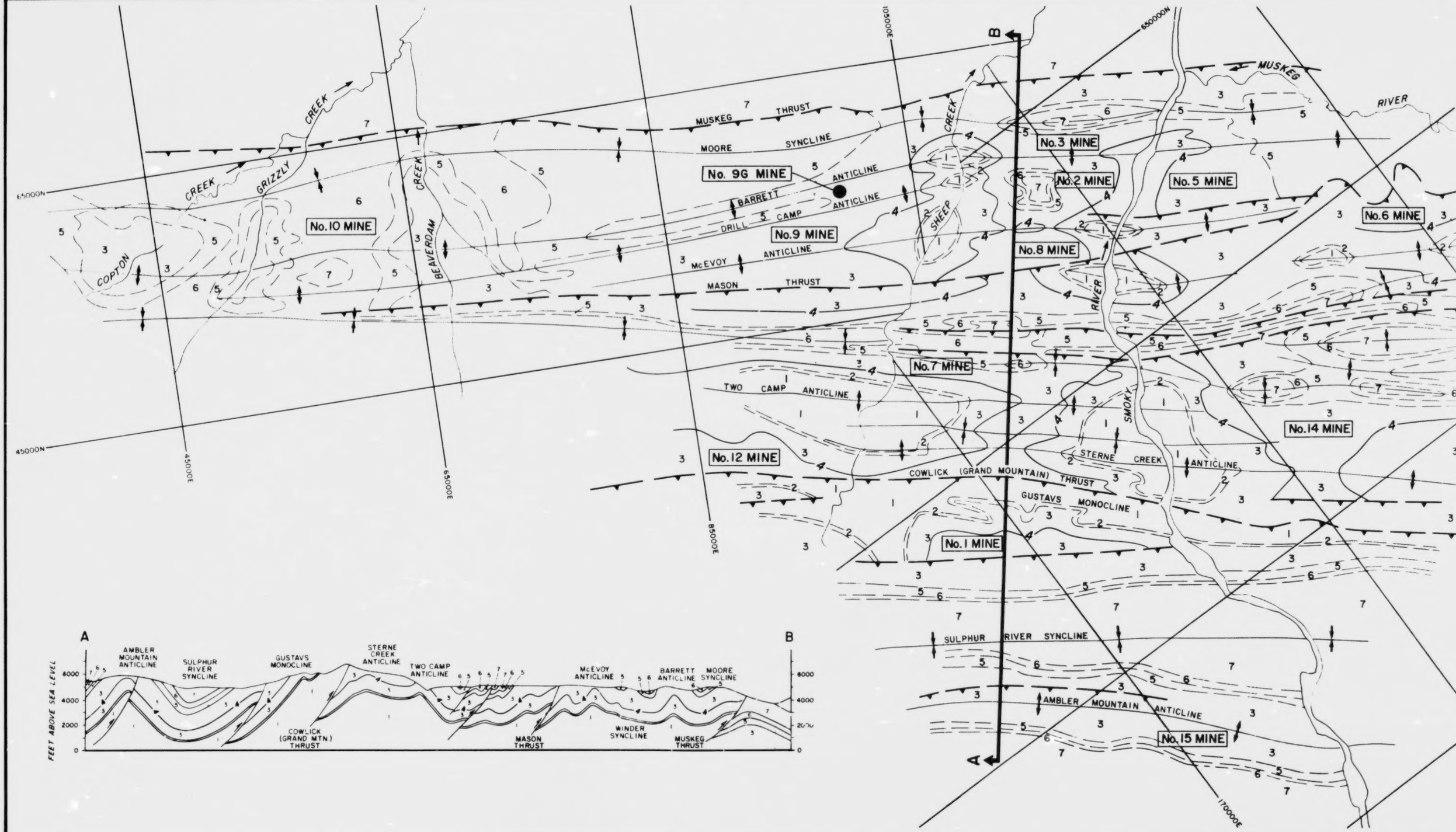
## COAL RESOURCES*
(000's Long Tons)

| Mine Area | Seam | SURFACE S.G. 1.35 Measured | SURFACE S.G. 1.35 Indicated | UNDERGROUND S.G. 1.38 MODERATE DIP Measured | MODERATE DIP Indicated | MODERATE DIP Inferred | STEEP DIP Measured | STEEP DIP Indicated | STEEP DIP Inferred |
|---|---|---|---|---|---|---|---|---|---|
| 1 | # 10 | | | | | | | 5,800 | 21,200 |
| | # 4 | | | | | | 26,800 | 16,000 | 24,800 |
| | # 2/3/5/7 | | | | | | | 64,550 | 37,000 |
| 2 | # 11 | | | 2,650 | | | | | |
| | # 10 | | | | 7,400 | | | | |
| 2-Barrett | # 11 | | | | | | 1,200 | | |
| | # 10 | | | | | | 1,400 | | |
| | # 4 | | | | | | 3,700 | | |
| 2-Reiff | # 11 | | | 1,180 | | | | | |
| | # 10 | | | 1,500 | | | | | |
| | # 4 | | | 1,630 | | | | | |
| 2-Sheep | # 11 | | | 4,400 | | | | | |
| | # 10 | | | 8,000 | | | | | |
| | # 4 | | | 4,400 | | | | | |
| 3 | # 11 | | | | | | | | 14,400 |
| | # 10 | | | | | | | 1,340 | 10,400 |
| | # 4 | | | | | | 11,700 | 16,400 | 10,400 |
| 5 | # 11 | | | | 5,760 | 14,000 | | | |
| | # 10 | | | 7,090 | 3,100 | 5,400 | | 1,070 | 4,500 |
| | # 4 | 1,150 | | | 4,000 | 17,400 | | | 30,300 |
| 9 | # 11 | 6,510 | | | | | | | 37,100 |
| | # 10 | 19,760 | | | | | | | 81,300 |
| | # 4 | | | 390 | | | | | 139,000 |
| 9A | # 11 | | | 3,220 | | | | | |
| | # 10** | | | 1,020 | | | | | |
| | # 4 | | | 6,000 | | | | | |
| 9B | # 11 | | | 840 | 860 | 2,000 | | | |
| | # 10** | | | 4,100 | 4,800 | | 530 | 440 | |
| | # 4 | | | 910 | | | | | |
| 9C | # 11 | | | 12,980 | 3,510 | | | | |
| | # 10** | | | | | | | | |
| | # 4 | | | | | | | | |
| 9G | # 11 | | | | | | | | |
| | # 10** | | | | | | | | |
| | # 4 | | | | | | | | |
| 7 | #4/6/7 | 6,420 | 4,750 | 8,330 | | | 2,700 | | |
| | # 4 | | | | | | | | |
| 14 | #3/4/7/10 | 11,980 | 8,010 | 16,350 | 10,570 | | | 71,690 | |
| TOTALS | # 11 | 1,150 | | 11,390 | 6,620 | 16,000 | 1,200 | | 51,500 |
| | # 10 | 6,510 | | 9,500 | 10,500 | 5,400 | 1,400 | 7,140 | 117,400 |
| | # 4 | 19,760 | | 47,750 | 12,310 | 17,400 | 45,430 | 33,910 | 204,500 |
| | Others | 18,400 | 12,760 | 16,350 | 10,570 | | | 136,240 | 37,000 |
| GRAND TOTALS | | 45,820 | 12,760 | 84,990 | 40,000 | 38,800 | 48,030 | 177,290 | 410,400 |

* The tonnages reported here do NOT represent all coal resources of the Smoky River coalfield, they are limited to those areas where some basis for evaluation has been established and an assessment of the resources has been made.

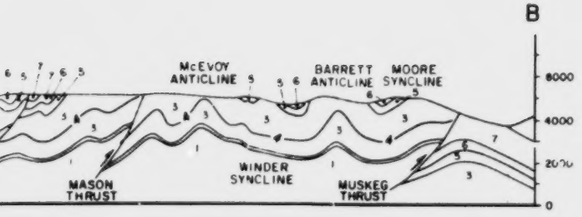
** Resource assessment not yet finalized, therefore excluded from this compilation.

McINTYRE MINES


MUSKEG THRUST
MOORE SYNCLINE
No. 9G MINE
ANTICLINE
BARRETT
DRILL CAMP
ANTICLINE
No.9 MINE
McEVOY ANTICLINE
MASON THRUST
No.10 MINE
No.12 MINE
No.7 MINE
TWO CAMP ANTICLINE
No.3 MINE
No.2 MINE
No.5 MINE
No.6 MINE
No.8 MINE
No.14 MINE
No.1 MINE
No.15 MINE
STERNE CREEK ANTICLINE
COWLICK (GRAND MOUNTAIN) THRUST
GUSTAVS MONOCLINE
SULPHUR RIVER SYNCLINE
AMBLER MOUNTAIN ANTICLINE
GRIZZLY CREEK
COPTON CREEK
BEAVERDAM CREEK
SHEEP CREEK
SMOKY RIVER
MUSKEG RIVER
65000N
45000N
45000E
300000E
650000E
1700000E
A
B
AMBLER MOUNTAIN ANTICLINE
SULPHUR RIVER SYNCLINE
GUSTAVS MONOCLINE
STERNE CREEK ANTICLINE
TWO CAMP ANTICLINE
McEVOY ANTICLINE
BARRETT ANTICLINE
MOORE SYNCLINE
COWLICK (GRAND MTN.) THRUST
MASON THRUST
WINDER SYNCLINE
MUSKEG THRUST
FEET ABOVE SEA LEVEL
6000
4000
2000
0

MUSKEG THRUST
MOORE SYNCLINE
No. 9G MINE
ANTICLINE
BARRETT
DRILL CAMP ANTICLINE
No.9 MINE
McEVOY ANTICLINE
MASON THRUST
No.7 MINE
TWO CAMP ANTICLINE
No.12 MINE
COWLICK (GRAND MOUNTAIN) THRUST
GUSTAVS MONOCLINE
No.1 MINE
SULPHUR RIVER SYNCLINE
AMBLER MOUNTAIN ANTICLINE
No.15 MINE
SHEEP CREEK
No.3 MINE
No.2 MINE
No.8 MINE
No.5 MINE
No.6 MINE
No.14 MINE
SMOKY RIVER
STERNE CREEK ANTICLINE
MUSKEG RIVER
PEAVINE LAKE
GRAND CACHE LAKE
VICTOR LAKE
A
B
300000E
850000E
1700000E
1900000E
2100000E
5900000N
6100000N
6300000N
6500000N

B




## LEGEND

**UPPER CRETACEOUS**

7 — KASKAPAU FORMATION: SHALE, SANDY SHALE, MINOR IRONSTONE, SANDSTONE (MARINE)

6 — DUNVEGAN FORMATION: SANDSTONE, SANDY SHALE (MARINE & ALLUVIAL)

**LOWER CRETACEOUS**

5 — FORT ST. JOHN GROUP: SHALE, SILTY SHALE, MINOR SANDSTONE (MARINE)

3 4 3 — LUSCAR FORMATION: SANDSTONE, SHALE, CARBONACEOUS SHALE, MINOR CONGLOMERATE, COAL SEAMS (ALLUVIAL) #4 SEAM SHOWN

2 — CADOMIN FORMATION: CONGLOMERATE, MINOR SANDSTONE (ALLUVIAL)

**LOWER CRETACEOUS AND UPPER JURASSIC**

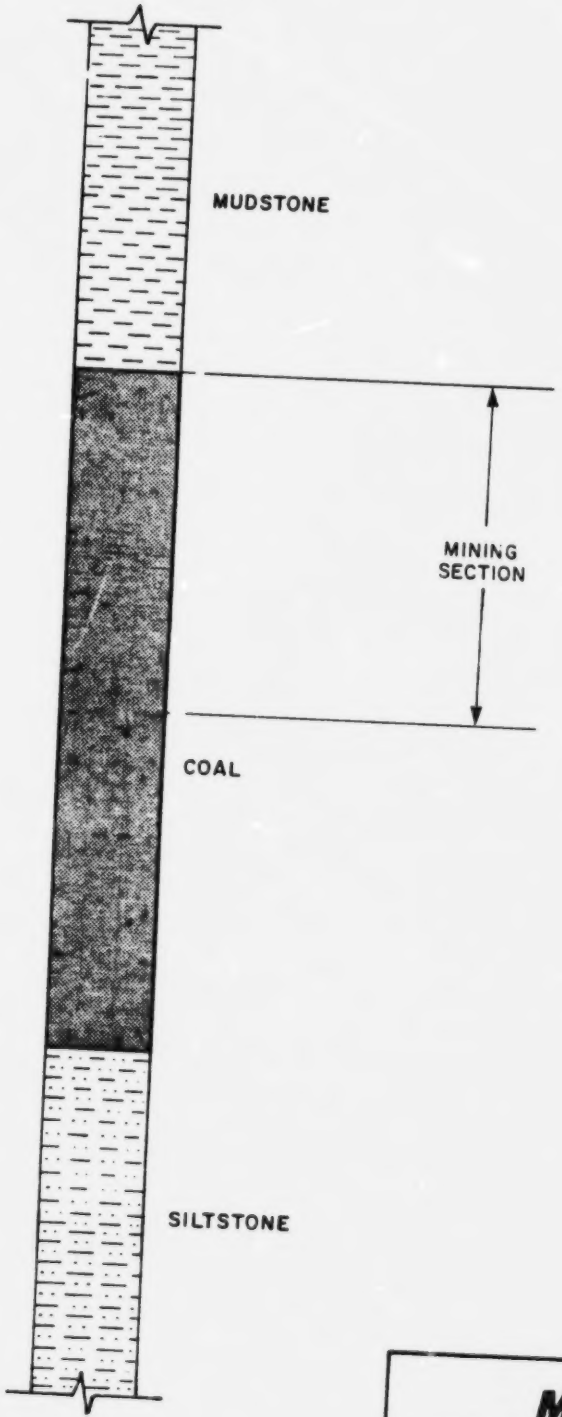
1 — NIKANASSIN FORMATION: SANDSTONE, SANDY SHALE, CARBONACEOUS SHALE, (MARINE AND ALLUVIAL)

**McINTYRE MINES**

# GEOLOGICAL SKETCH MAP OF THE SMOKY RIVER COAL FIELD

NOV. 1982 FIG. 1

0 5 000 10 000 15 000 20 000 feet
0 2 000 4 000 6 000 metres




DRILL HOLE 61-20

LOCATION . . . . . 2R-4 MINE

CO-ORDINATES . . . . . 645,010N
183,618E

COLLAR ELEVATION . . . 4573

DEPTH TO No. 4 SEAM . . 429 ft.

BORE INCLINATION . . . 90°

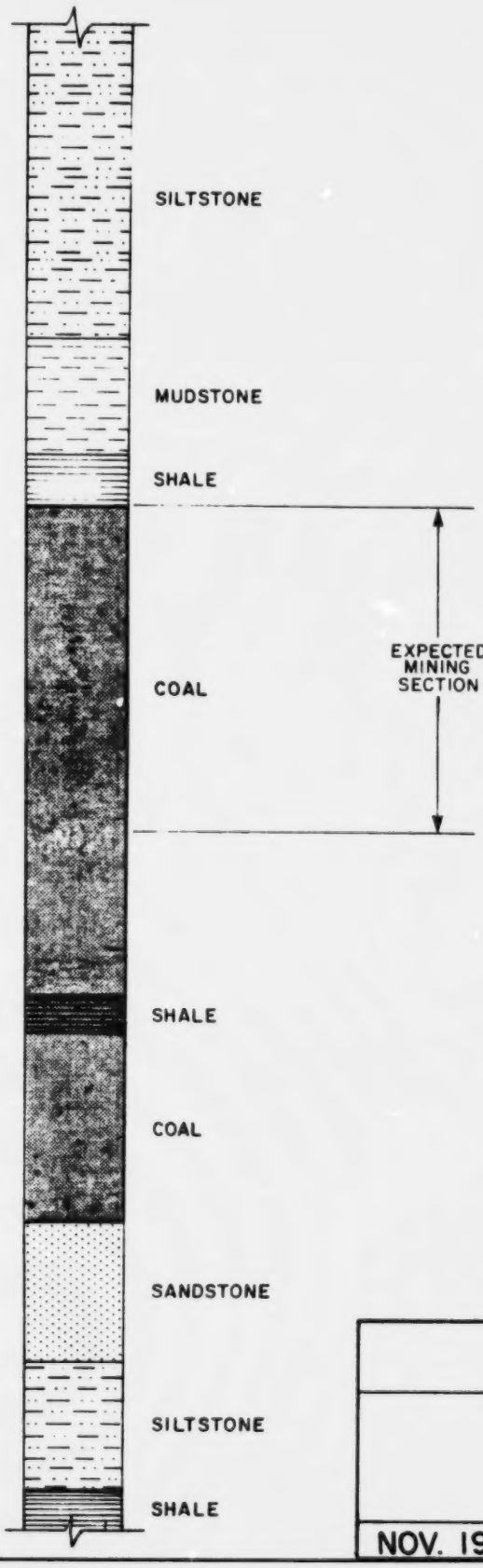
TOTAL DEPTH 610 ft.

SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

2R-4
SEAM STRATIGRAPHY

NOV. 1982 FIG. 2




DRILL HOLE 76-105

| | |
|---|---|
| LOCATION | No.9 G-4 MINE DRILL CAMP-ANTICLINE |
| CO-ORDINATES | 53,920.8N 104,479.2E |
| COLLAR ELEVATION | 4138.6 |
| DEPTH TO No. 4 SEAM | 523.5 ft. |
| BORE INCLINATION | 88.1°-S90W |
| TOTAL DEPTH | 563 ft. |

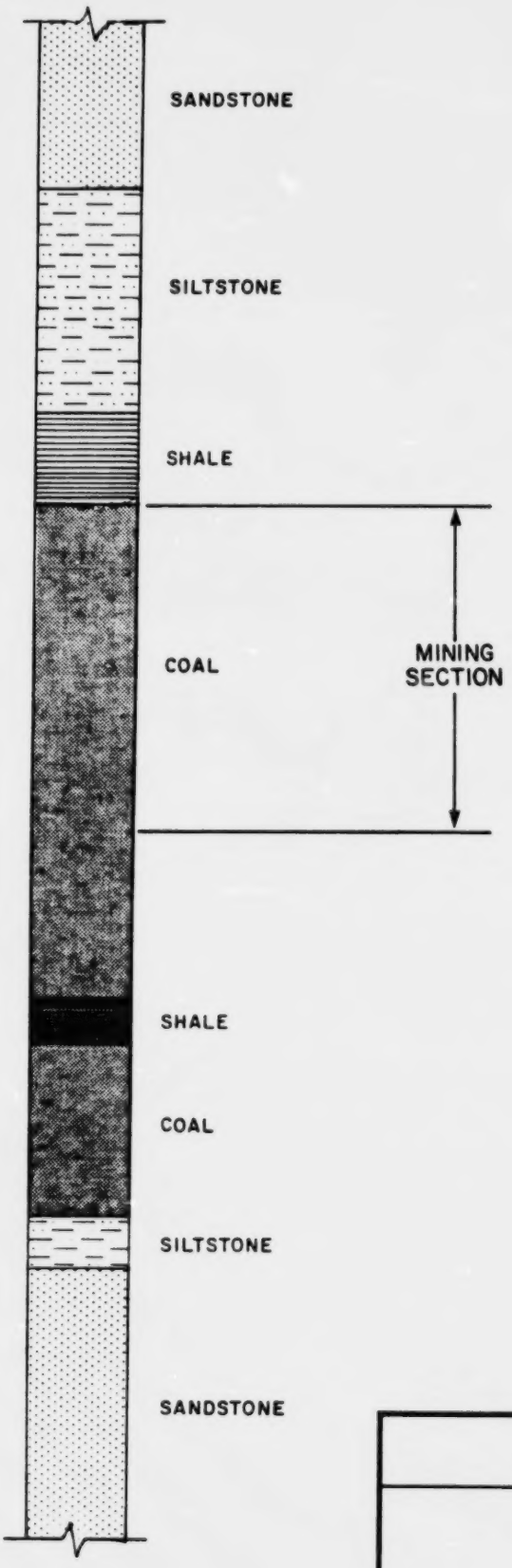
SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

9G-4

SEAM STRATIGRAPHY

NOV. 1982 FIG. 3




DRILL HOLE 76-02

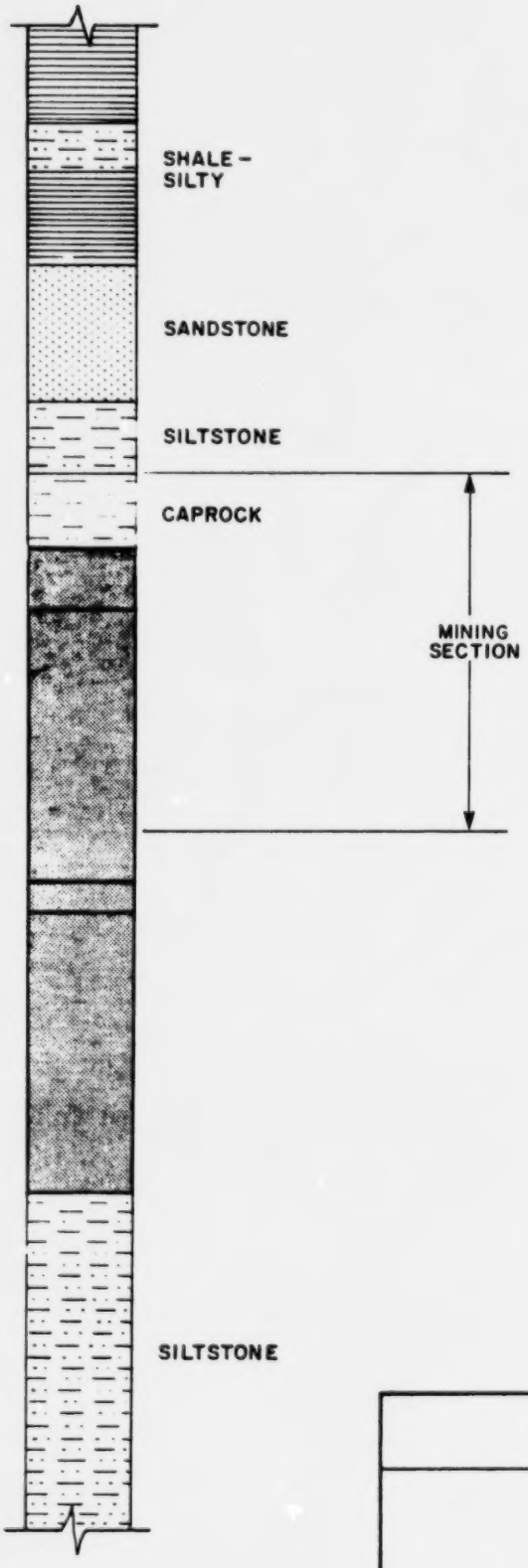
| | |
|---|---|
| LOCATION | 9A - 4 MINE |
| CO-ORDINATES | 51,172.9 N<br>94,129.2 E |
| COLLAR ELEVATION | 5442.0 |
| DEPTH TO No. 4 SEAM | 680.5 ft. |
| BORE INCLINATION | 82.8° - S42W |
| TOTAL DEPTH | 721 ft. |

SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

9A - 4
SEAM STRATIGRAPHY

NOV. 1982 FIG. 4




LITHOLOGICAL SECTION - RAILROAD CUT IN FRONT OF No. 5 MINE

LOCATION . . . . . 5 - 4 MINE

CO-ORDINATES . . . . . 639000N 197440E

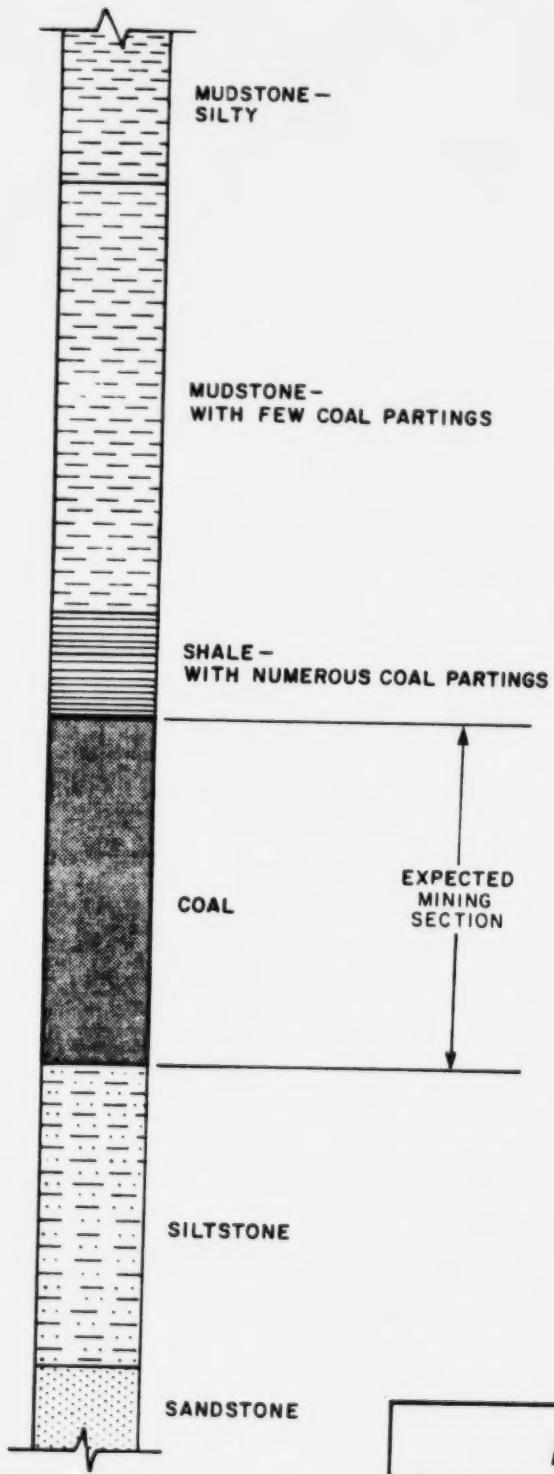
SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

5 - 4

SEAM STRATIGRAPHY

NOV. 1982 FIG. 5




LITHOLOGICAL SECTION
DRILL HOLE CC 11-7

LOCATION . . . . . 2-10 MINE

CO-ORDINATES . . . . . 644,780N
187,540E

COLLAR ELEVATION. . . 3650

DEPTH TO No.10 SEAM. . 94 ft.

BORE INCLINATION . . . -90°

BEDDING DIP . . . . . 15°

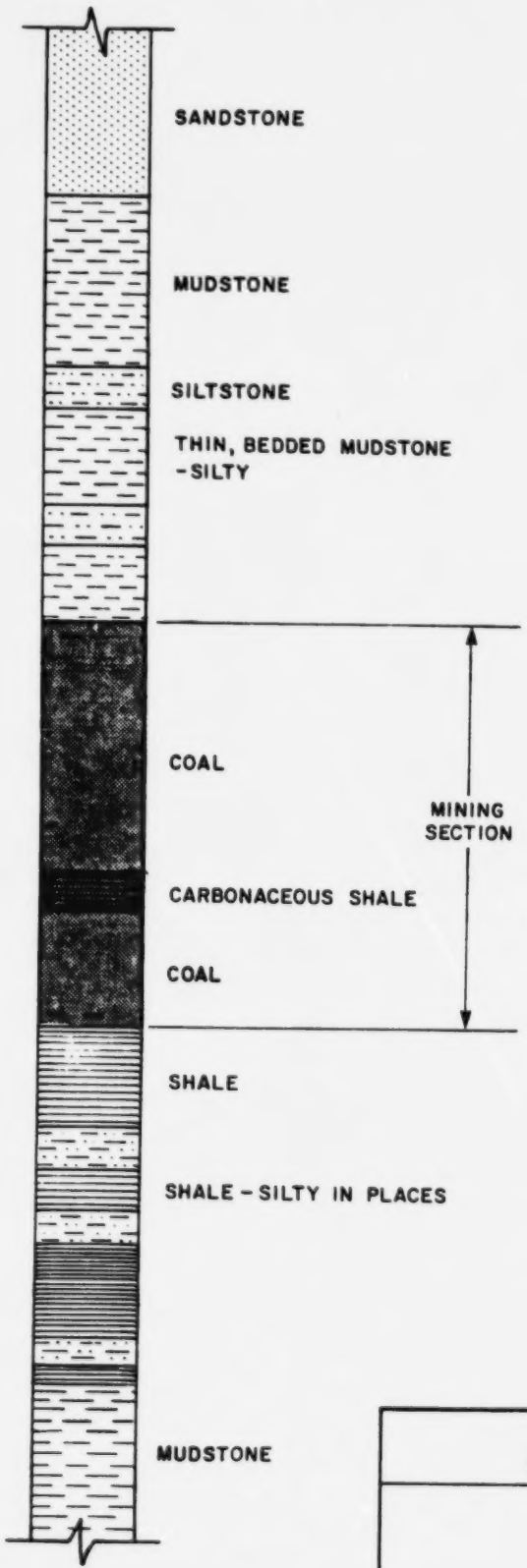
TOTAL DEPTH . . . . . 370 ft.

SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

2-10
SEAM STRATIGRAPHY

NOV. 1982 FIG. 6




DRILL HOLE 62 - 10E

LOCATION. . . . . 2 - 11 SMOKY

CO-ORDINATES . . . . . 645,635N
186,400E

COLLAR ELEVATION. . . 4484

DEPTH TO No. 11 SEAM. . 807 ft.

BORE INCLINATION. . . 90°

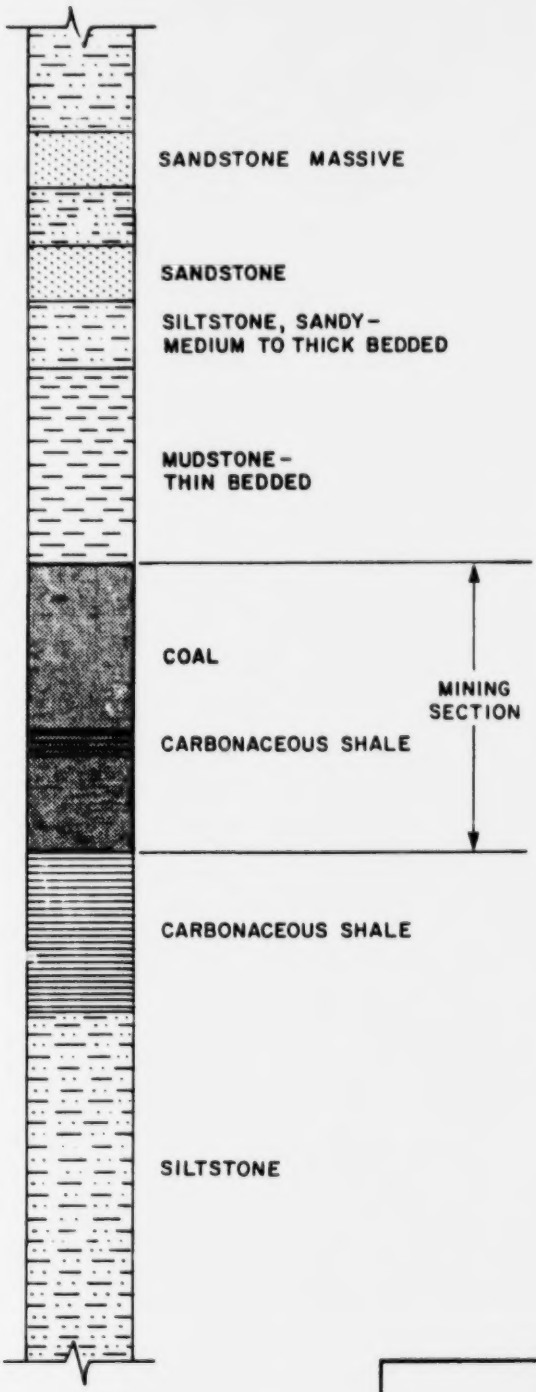
TOTAL DEPTH . . . . . 1225 ft.

SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

2 - 11
SEAM STRATIGRAPHY

NOV. 1982 FIG. 7




LITHOLOGICAL SECTION
OF No.II SEAM, No.I ENTRY

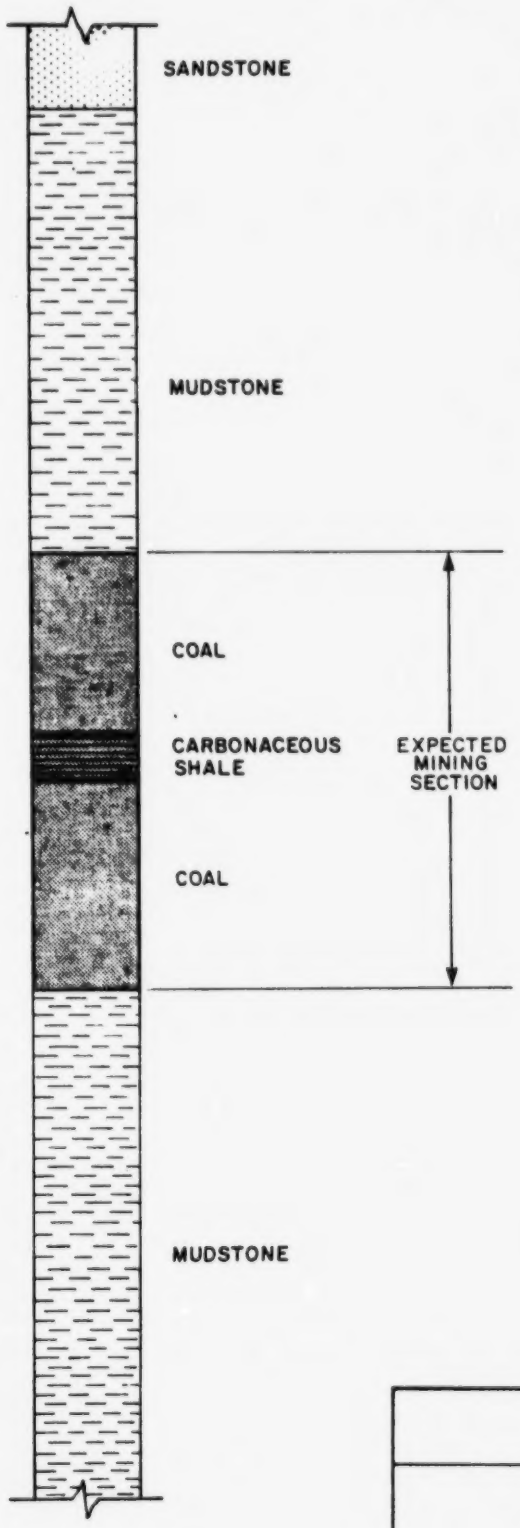
LOCATION . . . . . 2R-II MINE

SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

2R-II
SEAM STRATIGRAPHY

NOV. 1982 FIG. 8




LITHOLOGICAL SECTION–
RAILROAD CUT IN FRONT
OF No. 5 MINE

LOCATION. . . . . 5–11 MINE

SCALE: 1" = 5' VERTICAL

**McINTYRE MINES**

5 - 11

SEAM STRATIGRAPHY

NOV. 1982 FIG. 9

# APPENDIX 2-3

## ERCB GUIDELINES FOR INSTALLATION AND MAINTENANCE OF FIREFIGHTING ARRANGEMENTS

CODE OF PRACTICE

## 1 FOREWORD

This Code of Practice outlines the requirements for personnel and equipment to combat an outbreak of fire in underground mines. It establishes minimum standards and it is not intended to discourage or prohibit consideration of additional safety features or improved techniques.

## 2 UNDERGROUND FIRE FIGHTING

To avoid waste of coal which could result from an undergound fire, the following minimum requirements shall be met.

1. Water Supply

1.1 To secure a water supply to the underground water mains, there shall be provided:

(a) a reservior or water tanks, with a capacity of not less than 100 cubic metres, located on the surface adjacent to, or readily transportable to, any one shaft, slope or entry giving access to the mine, or

(b) a pumping system with a main pump, standby pump and a sump of capacity not less than 100 cubic metres.

1.2 The reservoir, tanks or sump shall be supplied directly from a surface water main or other dependable source, at a rate of feed adequate to satisfy the requirements of sub-paragraph 1.4 below.

1.3 Where electric pumps are required to secure the water supply, a standby pumping system whose prime mover is not dependent on the mine electrical system shall be provided.

1.4 Shaft, slope or entry mains, and underground water mains, shall be provided to supply water directly from the surface reservoir or tanks or the pumping system, and shall be of such diameter as to ensure a rate of flow of not less than 3.3 litres per second at the far end of the range.

2. Hydrants

Hydrants shall be provided

2.1 not more than 20 metres on the intake side of conveyor loading and transfer points, main junctions, and substations;

2.2 elsewhere along fire ranges such that the distances apart do not exceed 100 metres; and

2.3 at suitable central points in room and pillar workings and in close proximity to each longwall working face.

3. Fire Stations

3.1 A properly constructed fire station shall be provided in close proximity to the downcast shaft or slope bottom serving working districts, but in the case of a drift mine served by walkable outlets, the fire station may, with the approval of the Board, be on the surface.

3.2 The minimum equipment in or at such fire stations shall be:

(a) not less than 120 metres of fire hose, complete with couplings;

(b) a reserve supply of portable fire extinquishers;

(c) an adequate supply of sand and sand bags or an equivalent acceptable to the Board.

4. Fire Boxes

4.1 Fire boxes shall be set up in close proximity to the hydrants provided in accordance with paragraph 2 and at intervals along the mine roadway between the hydrants.

4.2 In each such box there shall be provided

(a) a branch pipe and nozzle of minimum 40-mm inlet diameter;

(b) sufficient hoses, each at least 15 metres long, to cover the distance from one fire box to the next in line; and

(c) accessories which shall include a hydrant gate and, where the hoses used exceed 40 mm in diameter, a hose clamp and two hose wrenches.

5. Fire Hose

5.1 Fire hose shall be a minimum of 40-mm diameter and designed to withstand a working pressure of 1000 kilopascals.

5.2 Fire hose shall be of lengths of not less than 15 metres (when new) and fitted with approved couplings.

5.3 Such fire hose shall be kept in fire boxes, recesses or compartments especially for that purpose.

6. Portable Extinguishers

At least two suitable fire extinguishers (in addition to a supply of stone-dust or sand) shall be provided on the intake side of all places where stationary electric motors, transformers or switch gear are in use.

7. Periodic Tests

Notwithstanding the generality of Division 10 of Part 1 of the Coal Mines Safety Regulations, the following tests and examinations shall be made at intervals not exceeding one month.

7.1 All hydrants shall be operated, and the hydrant at the most distant inbye end of the range tested for both quantity and pressure to ensure its adequacy.

7.2 Hoses, couplings, nozzles and other attachments shall be examined.

7.3 Fire extinguishers shall be examined.

A written report of all tests and examinations shall be countersigned by the assistant manager and deficiencies brought to the attention of the manager.

8. Fireproofing of Roadways at Conveyor Transfer or Loading Points

8.1 At each place where a conveyor transfer or loading point is installed, all parts of the roadway support and lining extending from at least 5 metres on the intake side to at least 10 metres on the return side of the complete installation shall be constructed of fire resistant materials or materials having a fire resistant coating approved by the Board.

8.2 There shall be a clearance between the bottom rollers of roadway conveyors and the floor of the roadway to permit removal of combustible material, and where this clearance is obtained by mounting the conveyor on pillars, the pillars shall be constructed of fire resistant material.

9. Training, Drills and Practices

9.1 Persons newly employed at a mine shall be given instruction in the use of fire fighting equipment during their initial period of employment.

9.2 All persons who work underground shall be given refresher training in the use of fire fighting equipment at intervals not exceeding two years.

9.3 The date on which each person received initial and refresher training shall be entered on his employment record.

9.4 Persons trained as instructors shall be appointed by the manager.

9.5 Fire drills shall be held in each section of the mine such that every person who works underground shall receive a fire drill at intervals not exceeding six calendar months.

9.6 All drills and practices shall be recorded by a person appointed by the manager, countersigned by the assistant manager, and any deficiencies brought to the attention of the manager.

10. Fire and Smoke Detection

The approval of the Board shall be obtained prior to the installation of any system of fire and smoke detectors or monitors.

11. Emergency Organization Plan

The Emergency Organization Plan required by Division 5 of Part 1 of the Coal Mines Safety Regulations shall include the following:

11.1 The scale shall be not less than 1:2500 and show a scale bar and north point.

11.2 The position of all fire ranges, hydrants, valves, fire stations and fire boxes shall be shown on the plan.

11.3 The plan shall be up-dated at intervals of not more than three months.

APPENDIX 2-4

TECHNICAL PAPER - THE
SUPPORT RESISTANCE REQUIRED BY LONGWALL FACES
BY A.H. WILSON

NATIONAL COAL BOARD

MINING RESEARCH AND DEVELOPMENT ESTABLISHMENT

# The Support Resistance required on Longwall Faces

by

A H Wilson

Head Rock and Coal Mechanics Branch

## Introduction

The ideal support resistance required is very difficult to determine by underground measurement. A certain amount of convergence on a longwall face is inevitable and will occur no matter how great the thrust of the supports. This irresistible convergence will produce a high load on a support which does not readily yield, but a low load on an inferior support or a support with loose uncompacted material above or below it. Differing degrees of compaction of loose material, floor penetration, failure of roof near the waste edge, leakage in the hydraulic system, varying supply pressure, all of these can cause differences of load from one support to another. Thus it is doubtful if the actual measured load to which a support is subjected underground can be used as a guide to the ideal load required to keep the roof in good condition.

In theory the ideal load can be obtained by gradually reducing the support resistance until roof instability occurs, evidenced by a break up of the lower roof or by an excessive increase in convergence. In practice this could be both dangerous and expensive, and is not recommended. Occasionally such conditions are reached adventitiously, and their occurrence has lead to the postulation of certain minimum mean load densities below which it is recognised that difficulties may be experienced. These give no indication, however, of the required distribution of resistance between the front and back legs, nor how the load should be varied for extreme conditions such as very thick seams or very wide webs.

## Amount of Rock Requiring Support

One approach to the problem is to consider the mass of rock which must be held in position by the supports. Figure 1 shows a cross section through a typical longwall face. The roof has been divided into two zones; the lower roof which caves in the waste and the upper roof which, although it may develop fractures, remains continuous and gradually lowers, compressing the caved material below it.

The horizontal stresses originally present in ground help maintain the stability of the upper roof, any joints or breaks being closed by the compressional forces. This effect is augumented over the working area by the concave-down curvature of the roof which adds to the compression on the lower surface of the upper roof. The lower roof is not continuous, and has a free face along the caving line. The horizontal compressional forces will be absent, and it must rely for stability on the supports. Should the supports be removed, the lower roof will collapse and the caving line will move forward to the face. The upper roof, however, will still remain intact, and hence it is only the lower roof which need be considered when assessing the required support thrust.

If the roof is unbroken over the working area, the supports will not have to take the full weight of the lower roof, as it will possess a certain degree of inherent stability as a cantilever projecting out from the face, and also possibly from adhesion between the beds of the lower roof and the beds of the upper roof. Should a face break occur, however, combined with a parting between the upper and lower roof beds, then the full weight of the lower roof will be thrown on to the supports. Although not necessarily common, this possibility is present on every face, and it is this extreme condition which should be adopted when considering the support resistance required to maintain the stability of the lower roof.

Application to Level Seams

Figure 2 shows typical cross-sections of the lower roof and face of seams worked in level conditions. The existence of a parting between the lower roof and the upper roof has been assumed at the caving height c; also a face break has been assumed at an angle equal to the caving angle $\alpha$. Should this parting and face break occur, the supports must keep in place a freed block of the lower roof as shown. The block itself need not be solid; it could be broken but kept in place as a stack of loose material resting on the support canopy.

This roof block will be acted on by three sets of external forces. These are:

1. The body weight W, which may be taken as a single vertical force acting at the centre of the block, distance w from the face line.

2. The combined support resistance P, acting at a distance p from the face line. For all practical purposes it can be assumed that P acts normal to the roof.

3. A reaction R between the freed block and the remaining mass of the roof acting at a distance r from the face line. R can be zero if P and W act along the same line, but only then. In all other circumstances R must be present. If the line of P is behind W, R will act at or near the back of the block; if the line of P is in front of W, it will act at or near the front of the block.

As W and P are both vertical, R must also be vertical. If moments are taken about R, then it can be shown that $P = W.\frac{r - w}{r - p}$. P will be the resultant of a number of rows of props. If two rows of props are considered, producing resistance of F and B at distances f and b from the face line, then the minimum values of F and B can be calculated in order to keep the lower roof in position. This has been done[1] for typical roof conditions and support spacings existing in the British coalfields, and the values are shown in Table 1.

It has been recognised for some time that on longwall faces greater thicknesses of extraction require greater support resistances; also that if the roof is weak and caves easily the bulk of the thrust must be given by the front legs, whereas if the roof is strong and tends to hold up, the maximum thrust must come from the back legs. Thus the distribution of load given in Table 1 conforms to known facts.

Frequently the support resistance required is expressed as the average resistance per unit area of the roof to be supported, referred to as the "mean load density". Table 2 shows the values of mean load density deduced from Table 1. These are given both in British nomenclature ($MN/m^2$) and in German

nomenclature ($Mp/m^2$ expressed in terms of the height of extraction M metres). At high angles of caving, the load is thrown on the front legs; at low angles of caving the load is thrown on the back legs. Hence if a support unit is to have universal application, its performance must be based on a figure at least twice the minimum mean load density given in Table 2.

Early underground work by MRE[2] indicated that the minimum mean load density for seams in the range 0.7 m to 1.7 m should be about 0.05 $MN/m^2$. Ashwin et al[3] concluded that this figure should be increased proportionally with the height of extraction. Also German workers in the past have used a figure of 8 M $Mp/m^2$ in their design of supports, but for universal application they are now increasing this figure to 15 M $Mp/m^3$. Thus the mean load density figures deduced in Table 2 are in line with those experimentally obtained. This gives confidence to apply the described method of approach to special conditions in which past experience cannot be used as a guide.

One such recent application in Britain concerned the cutting of coal by wide-web extraction. On this particular face the normal height of extraction was 2.4 m, the depth of web 0.5 m ($\ell$ = 3.8 m) and the caving angle was approximately 90°. Under these conditions the calculated required resistance by the front legs was 0.66 MN per chock. It was desired to increase the depth of web to 1.0 m ($\ell$ = 4.3 m) without changing the type of support, which were designed to give front leg loadings of 0.80 MN per chock. The calculation showed that to accomplish this with safety the height of extraction would have to be reduced to 2.0 m. This was done, and measurements before and after the change-over showed that the excellent roof conditions had been maintained. The front legs, however, were now coming up to yield which they had not previously done, and measurements of change of roof slope showed that with the wide web the roof in advance of the front row of legs tended to dip down towards the face, particularly at the time of cutting. These were taken to indicate that, as predicted, the roof was close to the limit beyond which difficulty might have been experienced.

## Application to Inclined Seams

The stability of a postulated roof block can also be used to determine the degree of roof support required when mining inclined seams, as shown in figure 3. Again three forces have been considered; W the weight acting vertically down, P the combined support resistance assumed acting normal to the roof, and a reaction R between the block and the remaining mass of the roof.

A graphical solution is shown in (a) and (b) of figure 3. The equilibrium conditions require that the directions of the three forces meet in a point, and that the force diagram (b) forms a closed figure. As P and W do not act in the same direction, R must be inclined to the top of the roof block. If slippage between the block and the upper roof is to be prevented, the angle $\Theta$ must be less than the angle of friction $\phi$. The minimum values of P and R will be obtained when $\Theta = \phi$. An alternative method of solution by equating components and taking moments is shown in figure 3 (c).

From both these solutions it can be shown[1] that in order to prevent the block moving bodily in any direction,

$$P \text{ must be greater than } W.\left(\frac{\sin\delta}{\tan\Theta} + \cos\delta\right) \qquad (1)$$

and in addition in order to prevent rotation,

$$P \text{ must be greater than } W.\frac{\frac{1}{2}c\sin\delta + (r-w)\cos\delta}{(r-p)} \qquad (2)$$

To prevent the block sliding out, the conditions of equation (1) must be met at all time, even when the supports are first set. Thus this equation can be used to calculate the minimum setting load. Should the block merely rotate in either direction, this will cause movement against the force P, and because of the rising support characteristic the value of P will increase until further rotation is prevented - provided the yield load has not been exceeded. Thus equation (2) can be used to deduce the required yield load. Even during rotation, the laws of friction will apply, preventing the block sliding out. Although undesirable, such a rotation will not necessarily create a hazard.

As mentioned above, the minimum value of P will be obtained when $\theta$ , the angle which R makes to the normal, is equal to $\phi$ , the angle of friction. A survey of available literature, supported by more extensive data on angle of internal friction of rock, indicates that a value of $\phi = 22^{\circ}$ ($\tan \phi = 0.4$) is a reasonable figure to accept as a minimum standard. This value was used to plot the variation of required setting load with seam inclination, shown in figure 4. Also shown for comparison is the value of mean setting load density recommended by the German Inspectorate[4] which was based on a similar equation derived by Jacobi[5].

So far we have considered only a face rising to the dip. Equation (1) applies equally to a dip running along the face as shown in figure 5, including blocks of the lower roof which extend over a number of supports. Provided the roof remains continuous in this direction, little trouble should be experienced, as each block of roof will be supported by the one below it. Should one portion of the roof be removed, however, by slipping back into the waste or falling out in any other way, the whole up-dip section is in jeopardy unless the support load is sufficient to clamp the lower roof in place. If the face has components of inclination both in the direction of advance and along the face, it is the full inclination (ie the true dip) which should be used in checking the roof stability in this context.

Calculations have been made[1] using equation (2) for the minimum yield loads required in typical British conditions, and these values are shown in Table 3. Again the required resistance per support of the front legs has been designated by F; the resistance of the back legs by B. Various caving angles $\alpha$ seam inclinations $\delta$ and heights of extraction $h$ have been considered. The yield loads of the powered supports at present used in Britain will cover most of the values shown, with the exception of the $90^{\circ}$ caving angle in the thicker seams. Here much greater thrust will be required from the front legs, and as r and b (the distances of R and B from the face) are alike in this situation, little or no advantage can be gained from increasing the thrust given by the back legs.

## Concluding Remarks

Despite the drawback that actual underground measurements do not supply information usable for deducing the ideal required support thrusts without endangering the face, it would appear that considering the roof in need of support to be a freed block, whose height and shape are governed by the manner of caving, does give information which conforms to known experience and measurements. The concept can be extended to cover conditions not covered by previous experience.

Although this report has concentrated on the frame type powered support, distributing the required resistance between two rows of legs, the concept can equally be applied to shield supports. In this case the overall support resistance P can be assumed concentrated at the hinge of the top canopy; otherwise the method of calculation remains the same.

This paper is presented by kind permission of Mr P G Tregeiles, Director of Mining Research and Development, National Coal Board. The opinions expressed by the author are his own and do not necessarily represent the views of the National Coal Board.

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The Mining Engineer, No. 173, Vol.134, June 1975, pp 479-491.

2 Wilson, A H. Conclusions from Recent Strata Control Measurements made by the Mining Research Establishment.
The Mining Engineer, No. 43, Vol.123, April 1964, pp 367-380.

3 Ashwin, D P, Campbell, S G, Kibble, J D, Haskayne, J D, Moore, J F A and Shepherd, R. Some Fundamental Aspects of Face Powered Support Design.
The Mining Engineer, No. 119, Vol.129, August 1970, pp 659-675.

4 Oberbergamt Dortmund. Richtlinien des Oberbergamts in Dortmund für die betriebsplanmässige Zulassung von Strebbruchbau von 26 Mai 1966. (G-Z 22/1 - 110.24/2907/66). Dortmund, 35 pp.

5 Jacobi, O. Die Ermittlung des Ausbauwiderstandes bei der Planung von Bruchbaustreben.
Glückauf, 100, 11 March 1964, pp 337-342.

## Table 1

## Minimum Values for F and B in Level Seams

(assuming c = 2h, f = $\ell$ - 1.6 m, b = $\ell$ - 0.5 m, r acts 0.5 m from the relevent corner, rock density = 0.025 MN/m$^3$, support spacing along face = 1 m).

| h | $\alpha$ | $\ell$ = 3.0 m | | $\ell$ = 3.6 m | | $\ell$ = 4.2 m | |
|---|---|---|---|---|---|---|---|
| | | F | B | F | B | F | B |
| m | deg | MN | MN | MN | MN | MN | MN |
| 1 | 90 | 0.14 | 0.01 | 0.21 | 0 | 0.31 | 0 |
| | 75 | 0.10 | 0.05 | 0.17 | 0.01 | 0.25 | 0 |
| | 60 | 0.06 | 0.09 | 0.12 | 0.06 | 0.19 | 0.02 |
| | 45 | 0 | 0.15 | 0.05 | 0.13 | 0.11 | 0.10 |
| | 30 | 0 | 0.21 | 0 | 0.21 | 0 | 0.22 |
| 2 | 90 | 0.27 | 0.03 | 0.43 | 0 | 0.61 | 0 |
| | 75 | 0.13 | 0.17 | 0.25 | 0.11 | 0.41 | 0.01 |
| | 60 | 0 | 0.32 | 0.05 | 0.31 | 0.17 | 0.25 |
| | 45 | 0 | 0.45 | 0 | 0.46 | 0 | 0.47 |
| | 30 | 0 | 0.67 | 0 | 0.66 | 0 | 0.66 |
| 3 | 90 | 0.41 | 0.04 | 0.64 | 0 | 0.92 | 0 |
| | 75 | 0.08 | 0.37 | 0.24 | 0.30 | 0.46 | 0.17 |
| | 60 | 0 | 0.61 | 0 | 0.63 | 0 | 0.66 |
| | 45 | 0 | 0.90 | 0 | 0.89 | 0 | 0.90 |
| | 30 | 0 | 1.39 | 0 | 1.35 | 0 | 1.34 |

## Table 2

## Mean Load Densities for Level Seams

(Deduced from Table 1)

| h | $\alpha$ | Mean Load Density (UK nomenclature) | | | Mean Load Density (German nomenclature)* | | |
|---|---|---|---|---|---|---|---|
| | | $\ell$ = 3.0 m | $\ell$ = 3.6 m | $\ell$ = 4.2 m | $\ell$ = 3.0 | $\ell$ =3.6 m | $\ell$ = 4.2 m |
| m | deg | MN/m$^2$ | MN/m$^2$ | MN/m$^2$ | Mp/m$^2$ | Mp/m$^2$ | Mp/m$^2$ |
| 1 | 90 | 0.05 | 0.06 | 0.07 | 5 M | 6 M | 7 M |
| | 75 | 0.05 | 0.05 | 0.06 | 5 M | 5 M | 6 M |
| | 60 | 0.05 | 0.05 | 0.05 | 5 M | 5 M | 5 M |
| | 45 | 0.05 | 0.05 | 0.05 | 5 M | 5 M | 5 M |
| | 30 | 0.07 | 0.06 | 0.05 | 7 M | 6 M | 5 M |
| 2 | 90 | 0.10 | 0.12 | 0.14 | 5 M | 6 M | 7 M |
| | 75 | 0.10 | 0.10 | 0.10 | 5 M | 5 M | 5 M |
| | 60 | 0.11 | 0.10 | 0.10 | 6 M | 5 M | 5 M |
| | 45 | 0.15 | 0.13 | 0.11 | 8 M | 7 M | 6 M |
| | 30 | 0.22 | 0.18 | 0.16 | 11 M | 9 M | 8 M |
| 3 | 90 | 0.15 | 0.18 | 0.22 | 5 M | 6 M | 7 M |
| | 75 | 0.15 | 0.15 | 0.15 | 5 M | 5 M | 5 M |
| | 60 | 0.20 | 0.17 | 0.16 | 7 M | 6 M | 5 M |
| | 45 | 0.30 | 0.25 | 0.22 | 10 M | 9 M | 7 M |
| | 30 | 0.46 | 0.38 | 0.32 | 16 M | 13 M | 11 M |

*M = height of extraction expressed in metres.

Note: As resistance of support may be required entirely on front legs or on back legs, to be universally applicable its design should be based on a figure equal to twice the minimum mean load density.

## Table 3

### Minimum Values for F and B in Inclined Seams

(assuming $\ell$ = 3.6 m, c = 2 h, f = $\ell$ - 1.6 m, b = $\ell$ - 0.5 m, r acts 0.5 m from the relevant corner, rock density = 0.025 MN/m$^3$, support spacing along face = 1 m)

| $\alpha$ | $\delta$ | h = 1 m | | h = 2 m | | h = 3 m | |
|---|---|---|---|---|---|---|---|
| | | F | B | F | B | F | B |
| deg | deg | MN | MN | MN | MN | MN | MN |
| 90 | 0 | 0.21 | 0 | 0.43 | 0 | 0.64 | 0 |
| | 10 | 0.24 | 0.02 | 0.53 | 0 | 0.88 | 0 |
| | 20 | 0.26 | 0.07 | 0.62 | 0.02 | 1.10 | 0 |
| | 30 | 0.27 | 0.11 | 0.69 | 0.07 | 1.29 | 0 |
| | 40 | 0.27 | 0.16 | 0.75 | 0.11 | 1.44 | 0 |
| 60 | 0 | 0.12 | 0.06 | 0.05 | 0.31 | 0 | 0.63 |
| | 10 | 0.09 | 0.17 | 0.17 | 0.34 | 0.17 | 0.60 |
| | 20 | 0.11 | 0.21 | 0.22 | 0.43 | 0.32 | 0.65 |
| | 30 | 0.13 | 0.25 | 0.25 | 0.51 | 0.38 | 0.76 |
| | 40 | 0.14 | 0.29 | 0.29 | 0.57 | 0.69 | 1.37 |
| 30 | 0 | 0 | 0.21 | 0 | 0.66 | 0 | 1.35 |
| | 10 | 0.09 | 0.15 | 0 | 0.69 | 0 | 1.43 |
| | 20 | 0.11 | 0.21 | 0 | 0.70 | 0 | 1.46 |

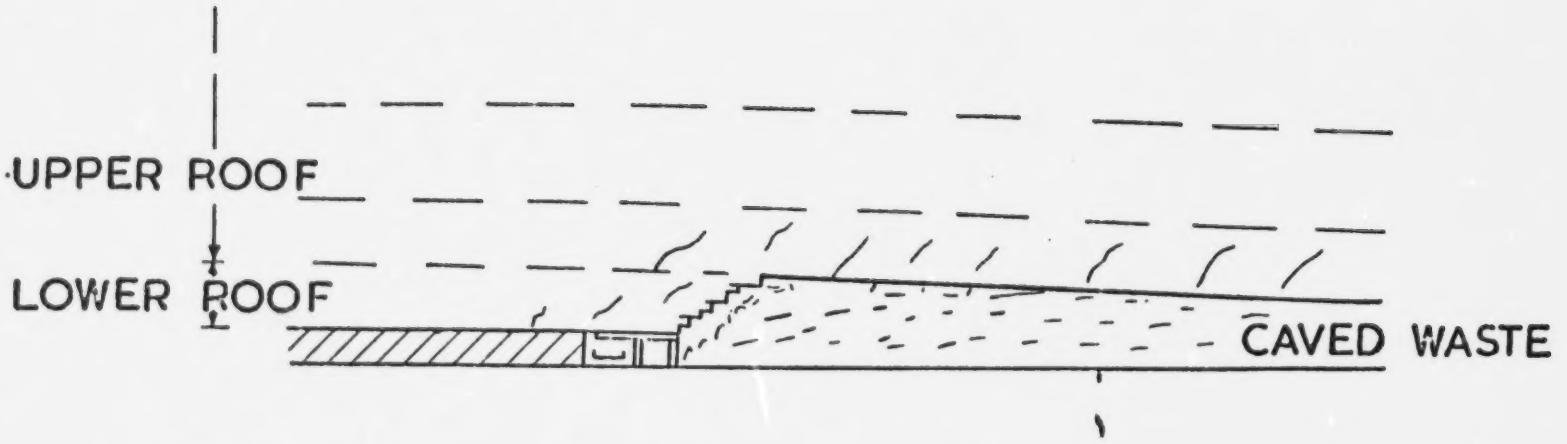
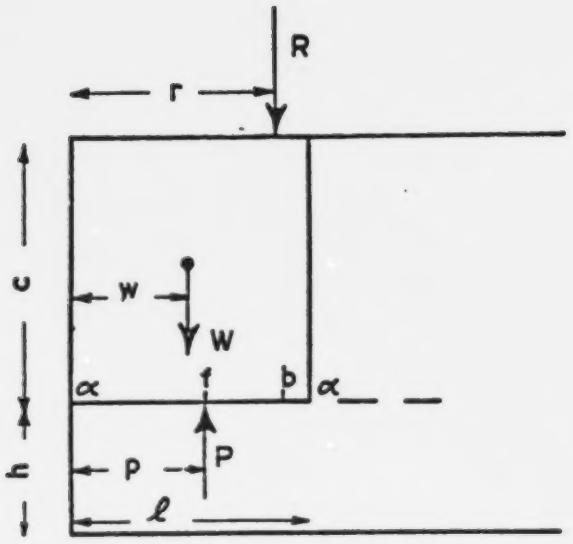
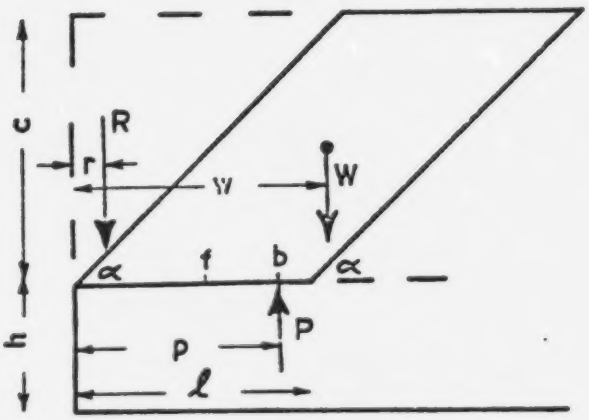



FIGURE 1. CROSS SECTION OF LONGWALL FACE SHOWING UPPER AND LOWER ROOF

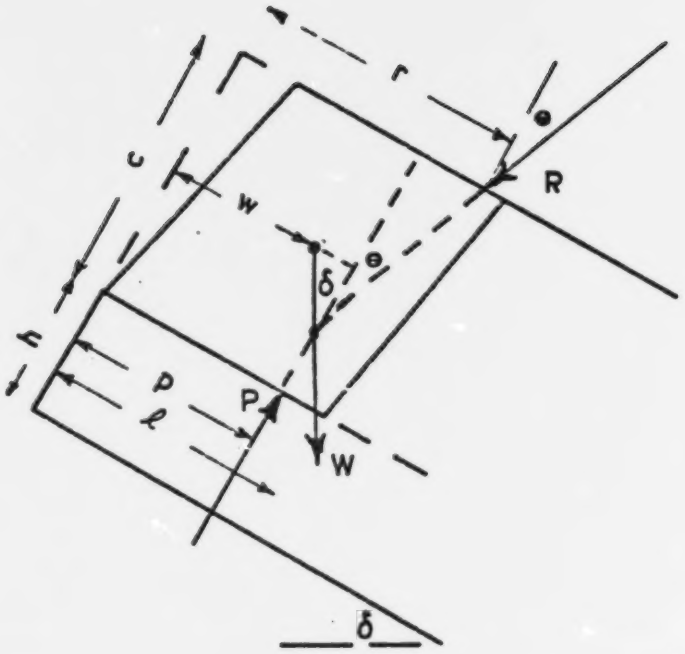



(a) Weight forward of support thrust

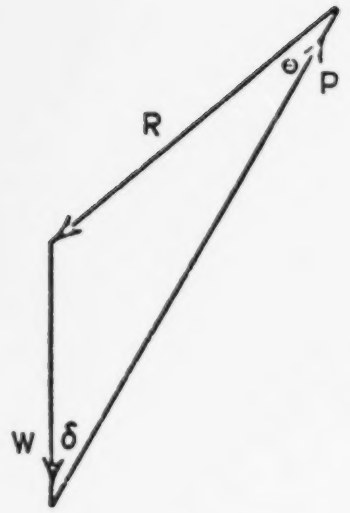



(b) Weight to rear of support thrust

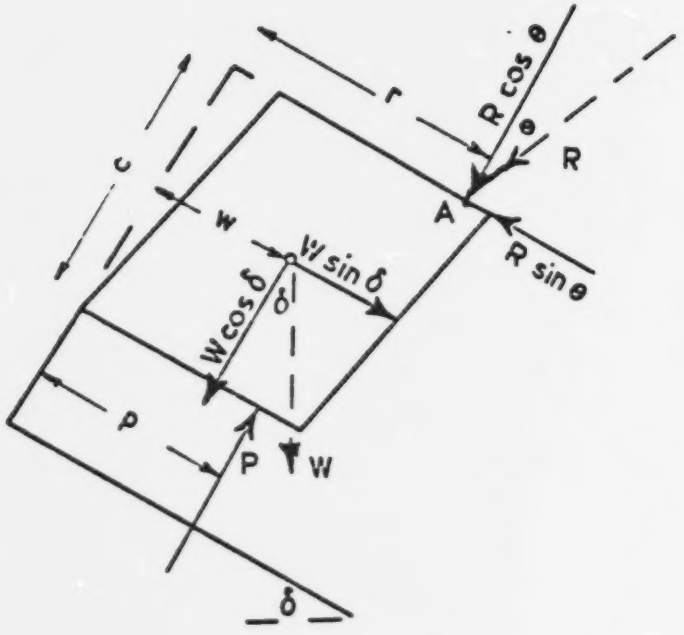
FIGURE 2. CONCEPT OF ROOF BLOCK AND FORCES INVOLVED (LEVEL SEAMS)




(a) Position of forces




(b) Corresponding force diagram




(c) Required equilibrium of force components

To prevent bodily movement:

$R \sin\theta = W \sin\delta$

and $P = W\cos\delta + R\cos\theta$

To prevent rotation:
moments about A

$P \times (r-p) = W\cos\delta \times (r-w) + W\sin\delta \times \frac{c}{2}$

FIGURE 3. CONCEPT OF ROOF BLOCK AND FORCES INVOLVED ( INCLINED SEAMS)

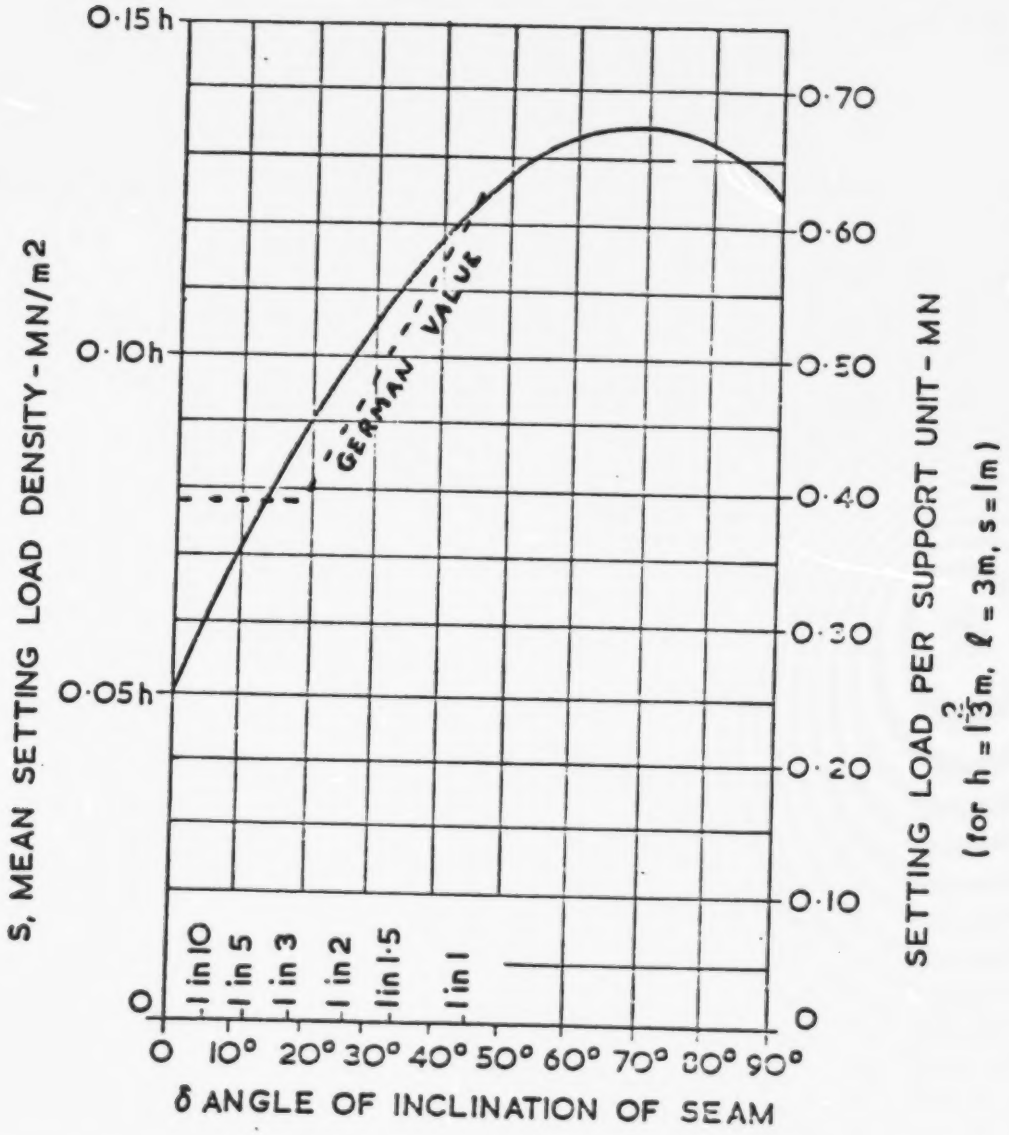



FIGURE 4. VARIATION OF REQUIRED SETTING LOAD WITH INCLINATION OF SEAM

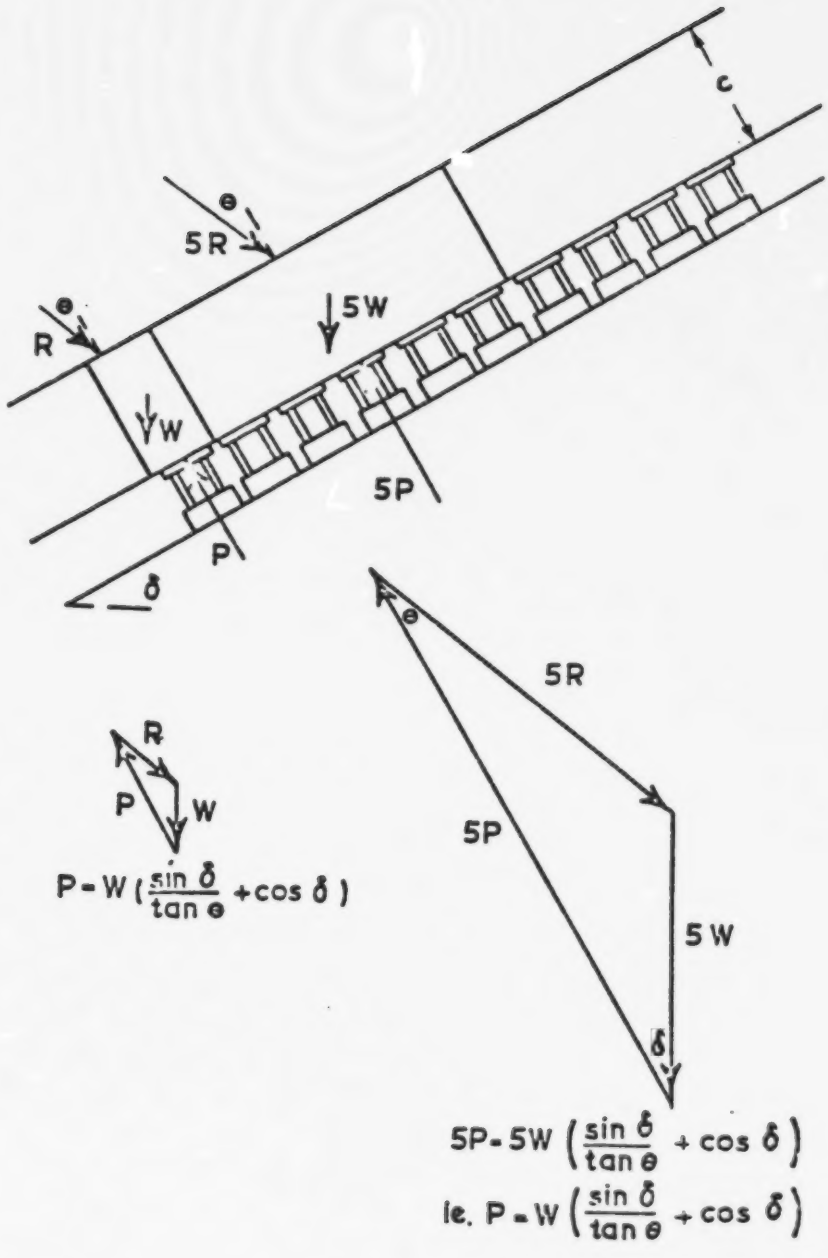



FIGURE 5. CROSS SECTION ALONG INCLINED FACE

# APPENDIX 2-5

# DETAILED CALCULATIONS

# CALCULATION OF LONGWALL SUPPORT CAPACITY

Refer to Figure 4-4. For the calculation of support capacity the following demensions have been used.

H = 40 ft

L = 14 ft

C = 5 ft centre to centre for chocks

d = 170 pcf density

a = angle of caving to vertical (40°)

W = total dead load on chock

$= L \times H \times C \times d$

$= \frac{14 \times 40 \times 5 \times 170}{2240} = 212.5$ tons

W = distance from face line

$= \frac{L}{2} + \frac{H}{2} \tan a$

$= \frac{14}{2} \times \frac{40}{2} \tan a$

= 23.8 ft

For equilibrium

$R \times r = W \times w$

R = Support Resistance in tons

r = distance from face line (11.82 ft for typical Dowty support)

$R = \frac{W \times w}{r} = \frac{212.5 \times 23.8}{11.82}$

= 427.87 tons

At the surface tank the water level devices will be used to monitor a drop in water level and initiate appropriate pumping action. A small drop in water level will start the well pump, a further drop in water level will start the underground pumps. A similar device will stop the supply to the surface tank when a maximum level has been attained.

At the underground storage tanks level devices will also be used to control pump functions. A high level and two low level controls will be employed.

If the surface tank is at its maximum level and the underground storage becomes full the pumps will start and continue to pump water to surface drainage until Low Level Device No. 1 is activated. Pumping will then cease until the high level device is reactivated.

If the surface tank level is below its maximum level and the water level in the underground storage is above the low level No. 2 position, the underground pumps will operate until either the high level device in the surface tank operates or until the low level 2 position in the underground sump is reached.

The location of the various level sensing devices will be choosen to minimize frequent stop start operation of the pumps and maximize utilization of available water storage capacity. Table 5-2 shows arrangement of level devices and provides the proposed control sequence.

# WATER REQUIREMENTS FOR FIREFIGHTING

## Introduction

To meet the requirements of firefighting, a storage facility for some 23 000 gallons of water is required at all times.

In order to ensure that an adequate water supply is available and to provide operating safeguards against failure of water supply pumps, etc., a tank will be installed which will provide sufficient water storage for two full mining shifts. This will require a storage tank of 100 000 gallon capacity, 60 000 gallons available for dust suppression and 40 000 for firefighting. A general arrangement of the proposed system is shown on Figures 5-24 and 5-25.

## Operation of the System

Following initial filling of the surface storage tank the mechanical arrangement of valves and piping to the pumps will ensure that deliberate action must be taken to obtain access to the water supply stored for firefighting purposes. For normal operations water will be obtained for dust suppression purposes from pump suction columns installed at an appropriate level above the firefighting suction columns in the tank (Figure 5-24). To maximize available water resources level controls will be employed to ensure that the available water storage facility is used to its maximum advantage.

Water level devices will be installed in the surface storage tank and at the underground pumping site.

# APPENDIX 2-6

## COMPUTER ECONOMIC MODEL

* This appendix is contained only in the Confidential Addendum Economic Analysis submitted to CANMET under separate cover.

# APPENDIX 2-7

## DETAILED COST ANALYSIS COMPUTER PRINTOUT

* This appendix is contained only in the Confidential Addendum Economic Analysis submitted to CANMET under separate cover.

# APPENDIX 2-8

# TYPICAL EQUIPMENT SPECIFICATIONS

LONGWALL STUDY

TABLE OF CONTENTS

APPENDIX 2-8 TYPICAL EQUIPMENT SPECIFICATIONS


| PAGE | |
|---|---|
| 1 | Cable - Type SHD |
| 2 | Switchgear |
| 3 | Cable - Type SHD - GC |
| 4 | Communications |
| 5 | Continuous Miner |
| 6 | Shuttle Car |
| 7 | Stamler Feeder - Breaker |
| 8 | Roof Bolter |
| 9 | Conveyor Systems |
| 10 | Dosco LH 1300 |

## Metric Units

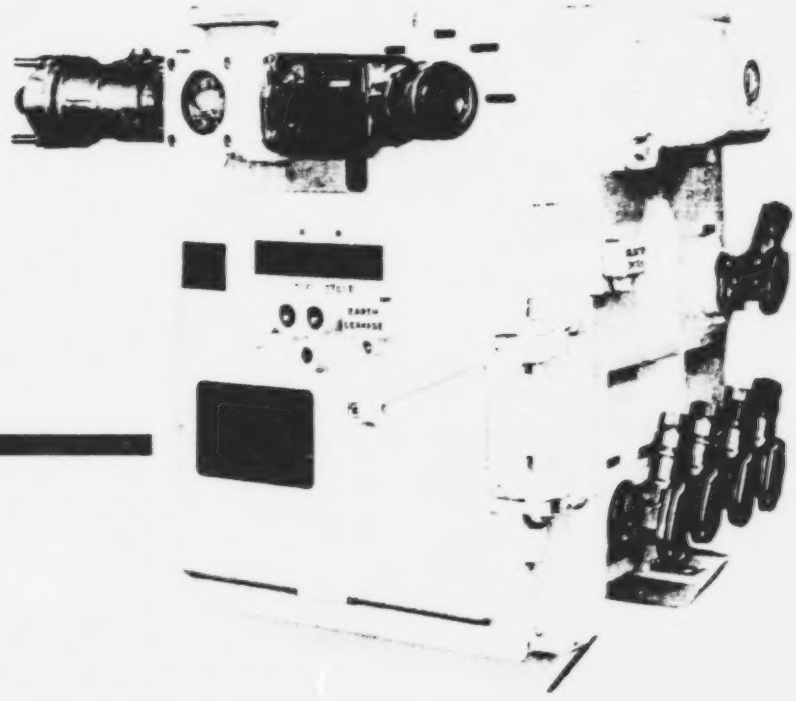
| Power Conductor Size, mm² | Insulation Thickness, mm | Ground Conductor Size, mm² | Jacket Thickness, mm | Nominal Overall Diameter, mm | Nominal Mass, kg/km | Ampacity, amperes |
|---|---|---|---|---|---|---|
| **0-2000 Volts** | | | | | | |
| 13 | 1.8 | 5.3 | 3.9 | 33 | 1540 | 93 |
| 21 | 1.8 | 8.4 | 3.9 | 36 | 1950 | 122 |
| 34 | 1.8 | 8.4 | 4.3 | 40 | 2565 | 159 |
| 42 | 2.0 | 11 | 4.8 | 45 | 3155 | 184 |
| 54 | 2.0 | 13 | 4.8 | 47 | 3670 | 211 |
| 67 | 2.0 | 17 | 5.2 | 51 | 4425 | 243 |
| 85 | 2.0 | 21 | 5.2 | 54 | 5185 | 279 |
| 07 | 2.0 | 27 | 5.6 | 59 | 6305 | 321 |
| 27 | 2.4 | 34 | 5.6 | 64 | 7515 | 355 |
| 73 | 2.4 | 42 | 6.0 | 71 | 9750 | 435 |
| 253 | 2.4 | 67 | 6.7 | 81 | 13395 | 536 |
| **2001-5000 Volts** | | | | | | |
| 13 | 2.8 | 5.3 | 4.7 | 40 | 2015 | 93 |
| 21 | 2.8 | 8.4 | 4.7 | 43 | 2480 | 122 |
| 34 | 2.8 | 8.4 | 5.2 | 47 | 3110 | 159 |
| 42 | 2.8 | 11 | 5.2 | 50 | 3600 | 184 |
| 54 | 2.8 | 13 | 5.6 | 53 | 4220 | 211 |
| 67 | 2.8 | 17 | 5.6 | 56 | 4890 | 243 |
| 85 | 2.8 | 21 | 6.0 | 60 | 5700 | 279 |
| 07 | 2.8 | 27 | 6.0 | 64 | 6875 | 321 |
| 27 | 3.0 | 34 | 6.4 | 68 | 8045 | 355 |
| 73 | 3.0 | 42 | 6.7 | 75 | 10155 | 435 |
| 253 | 3.0 | 67 | 7.1 | 84 | 13765 | 536 |

# Type SHD

**Three Power and Three Grounding Conductors**
**100% Insulation Level / Shielded**
## 0-25000 Volts

# Wallacetown

# Type A.67 1100 Volt Flameproof Gate End Box



**General**

This starter has been designed for the control and direct-on-line starting of squirrel cage induction motors on supply systems up to 1100 volts 50/60 hertz and is certified flameproof, certificate number FLP 5970.

**Current Ratings**

The unit is available in 150 amp, 250 amp and 300 amp ratings. The 300 amp unit can be supplied only with a vacuum contactor but either air break or vacuum contactors can be fitted for 150 or 250 amp ratings.

**The Wallacetown Engineering Company Limited Ayr U.K.**

## Metric Units

| Power Conductor Size, mm² | Insulation Thickness, mm | Ground Conductor Size, mm² | Ground Check Conductor Size, mm² | Jacket Thickness, mm | Nominal Overall Diameter, mm | Nominal Mass, kg/km | Ampacity, amperes |
|---|---|---|---|---|---|---|---|
| **0-2000 Volts** | | | | | | | |
| 13 | [illegible] | 53 | 5.5 | [illegible] | [illegible] | [illegible] | [illegible] |
| 21 | 1.8 | 84 | 8.4 | [illegible] | [illegible] | [illegible] | [illegible] |
| 34 | [illegible] | 13 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 42 | 2.0 | 17 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 54 | 2.0 | 21 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 67 | 2.0 | 27 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 85 | 2.0 | 34 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 107 | 2.0 | 42 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 127 | 2.4 | 54 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 173 | 2.4 | 67 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 253 | 2.4 | 107 | 8.4 | 6.7 | 81 | 13515 | 536 |
| **2001-5000 Volts** | | | | | | | |
| 13 | [illegible] | 53 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 21 | 2.8 | 8 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 34 | [illegible] | 13 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 42 | [illegible] | 17 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 54 | [illegible] | 21 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 67 | 2.8 | 27 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 85 | [illegible] | 34 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 107 | [illegible] | 42 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 127 | [illegible] | 54 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 173 | [illegible] | 67 | [illegible] | [illegible] | [illegible] | [illegible] | [illegible] |
| 253 | 3.0 | 107 | 8.4 | [illegible] | [illegible] | 13990 | 536 |

# Type SHD-GC

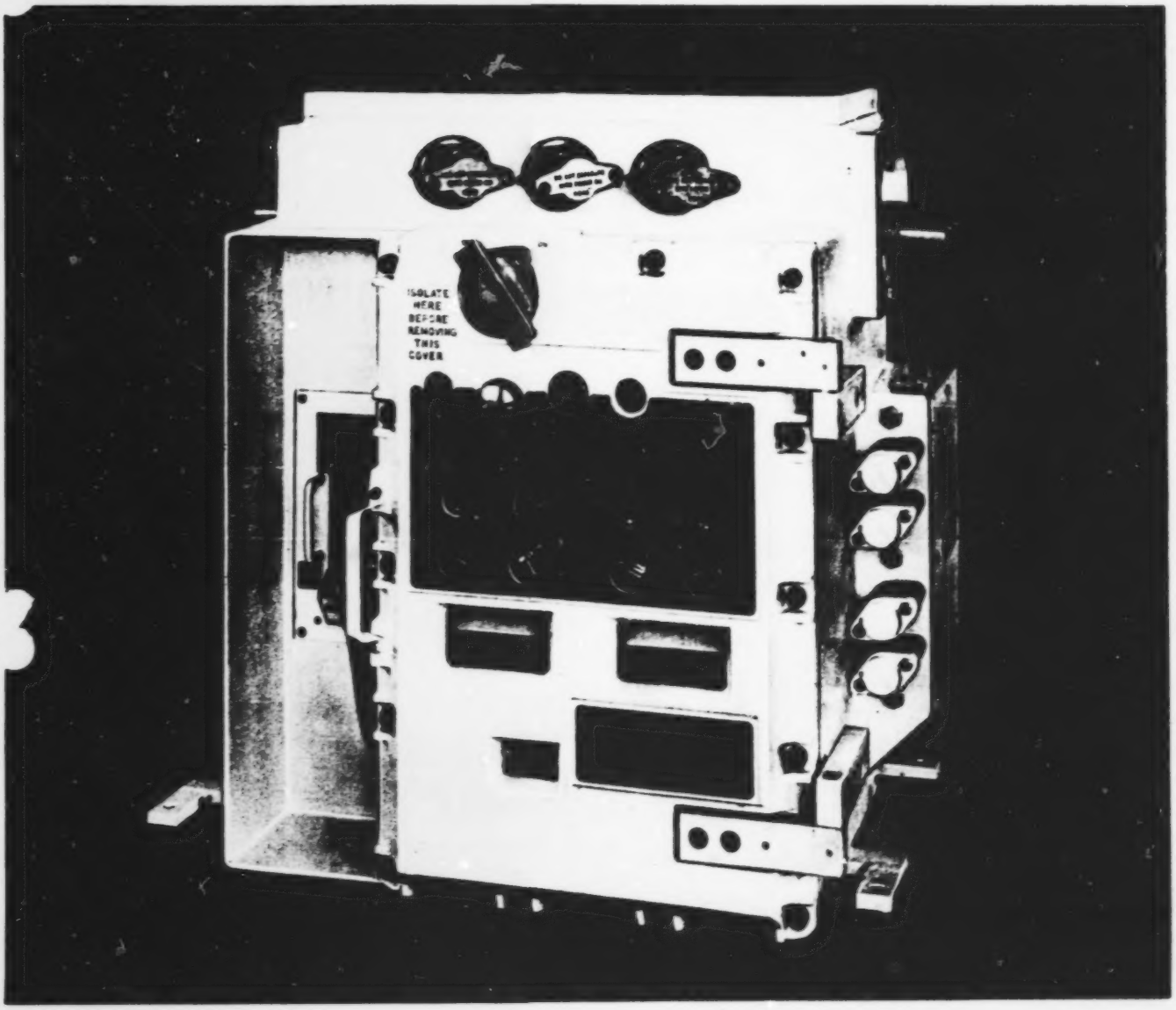
**Three Power, Two Grounding and One Ground Check Conductor**
**100% Insulation Level / Shielded**

## 0-25000 Volts

DD
DAVIS OF DERBY

# Mk II Sivad Conveyor Control & Communications System

member of the Royal Doulton Group




*** Increased Safety**

- Monitored pre-start warning for AFC & stage loader
- SIVAD systems operational in mines in the UK, Europe, South Africa, USA, and The Peoples Republic of China

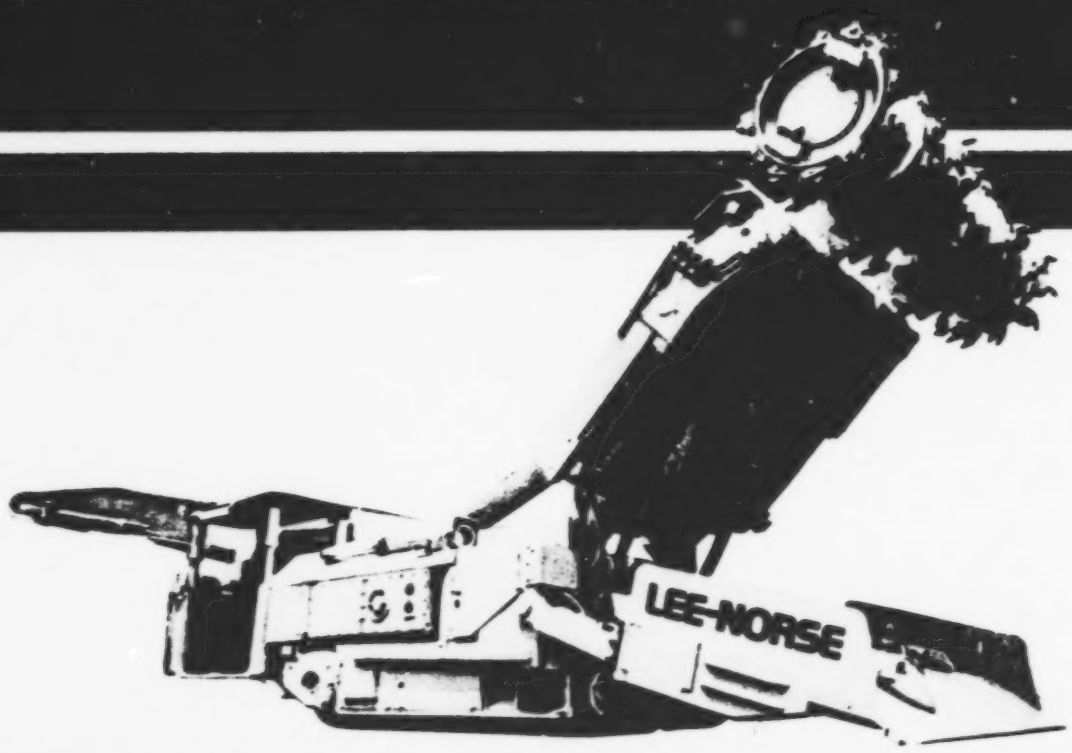
*** Reliable & Versatile**

- Selection of operating voltages from the range: 110V, 240V, 380V, 550V, 1000V, and 1100V.
- Suitable for P130 or BZ1/PR13 type pilot circuits

*** Ease of Maintenance**

- Circuit conditions and voltages indicated at the console
- Plug-in chassis for ease of replacement
- Maintenance manual and full training package available

# HH497-II
## Continuous Miner



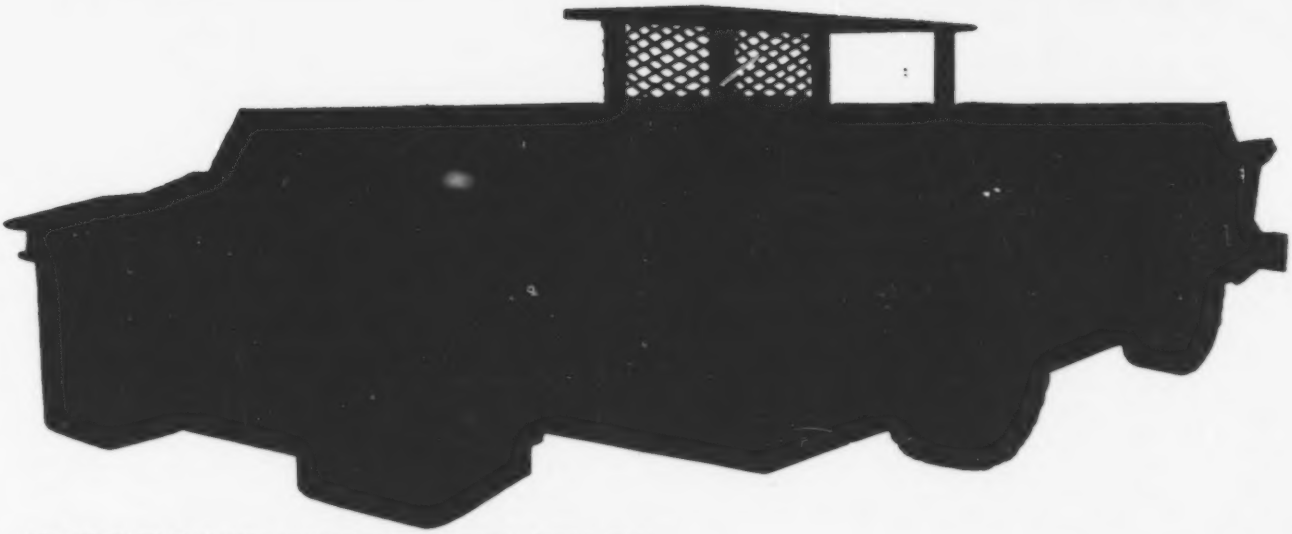

The HH497-II is a rugged 90,000 pound, narrow chassis model for mining in seams up to eleven-feet, six-inches. Design features make the HH-497-II ideal for operation on grades and in difficult floor conditions. Check the features listed below. You'll see why it's one tough miner.

- High 12¼" ground clearance
- Optimum bit lacing design by computer analysis provides:
  - Reduced shock in cutter drive train
  - Reduced stress at hinge points
  - Less vibration and greater operator comfort
- Accurate bit tip location assured by Lee-Norse electronic bit block positioning fixture
- Cutterhead with two-piece coupled cuttershaft allows:
  - Face seal replacement without cuttershaft disassembly
  - Independent disassembly of right or left cutter gear case
  - Complete cutterhead maintenance underground
- Compact cutterhead allows more space for better face ventilation
- Arm gathering head with face seals
  - Electric drive
  - Reversible conveyor
  - Stabilizers to minimize flexing
- Infinitely variable tram speed for cutting, tramming and loading
- Independent control of each crawler for positive traction
- Piano hinge-type crawler chain
- Lee-Norse integrated lighting system—MSHA approved
- Safety features:
  - Panic bar in operator's cab plus opposite side emergency stop
  - Water deluge system
  - Adjustable canopies certified to MSHA requirements
  - Two-hand start system for cutterhead and conveyor
  - Plastic laminated operator's manual secured in cab

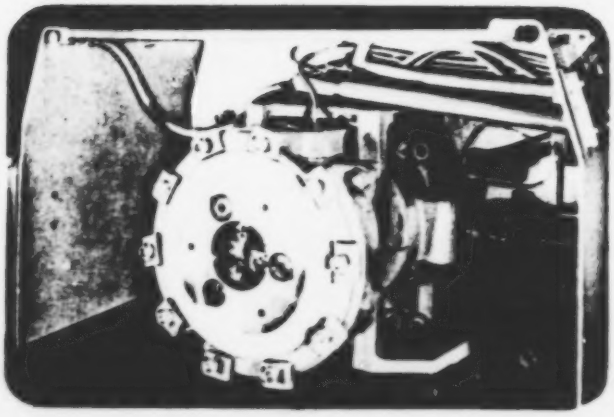
**LEE-NORSE**

Part of worldwide Ingersoll-Rand

# JOY 10SC22-40C 10SC22-48C 10SC22-56C 10SC22-64C SHUTTLE CARS



**The 10SC22-C shuttle car series is a rugged, dependable haulage vehicle available in 52½" height and in 40", 48", 56" and 64" conveyor widths. Two traction motors—either AC or DC—power a four-wheel positive drive that is guided by Orbitrol hydraulic steering for ease of handling and good maneuverability. Interchangeable wheel units have been strengthened with full planetary gearing to give a payload capacity of 12 tons. Conveyor speed is 100 FPM which discharges the load in 30 seconds. Standard equipment includes disc-type brakes, zone lubrication and 6", 9" or 12" sideboards.**



**Full planetary wheel units distribute load over three gears, thus affording greater strength and efficiency. Load-carrying capacity is 12 tons.**

## FEATURES:

- Stronger construction at lower cost
- Four-wheel positive drive
- Interchangeable heavy-duty wheel units employing full planetary gearing
- Hydraulic power steering
- Increased payload capacity
- Interchangeable traction and conveyor motors

# Stamler Feeder-Breakers

## High-Profile Models

For areas where ceiling height is not a limiting factor, the Stamler High Profile Feeder-Breaker for underground coal mining has greater throughput, more hopper capacity and can handle larger input sizes than low profile models.

The Stamler High Profile Feeder-Breaker accepts mined material from a front end loader, scoop, truck, ram car or LHD. The High Profile machine can work with haulage equipment with as much as 18 ton capacities.

### Standard Specifications

#### Frame

**Assembly:** All welded
**Materials:** Structural shapes and steel plate

#### Mobility

**Type:** Hydraulically driven crawlers
**Speed:** 40 FPM, hydraulic feeder-breaker
30 FPM, mechanical feeder-breaker

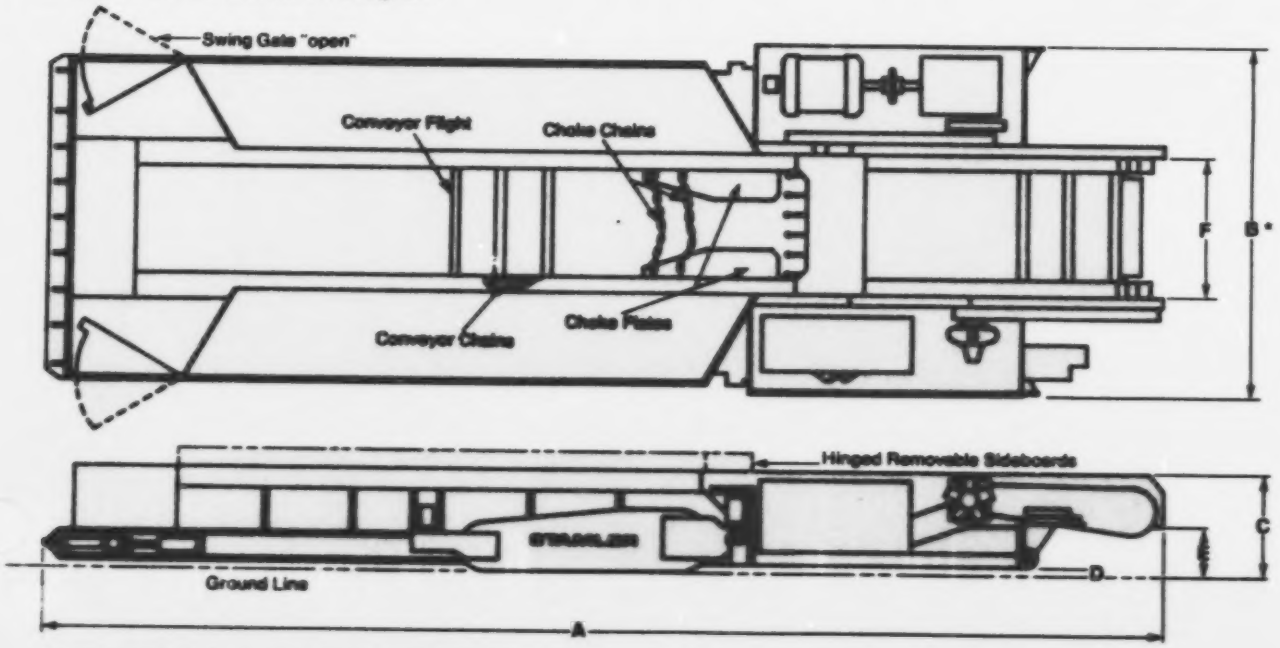
#### Electrical Data:

**Motors:**
Mechanical Drive Feeder-Breaker
– 2 MSHA approved motors.
Hydraulic Drive Feeder-Breaker
– 1 MSHA approved motor.

| Dimensions | Series 14 | Series 14-3 | Series 14-6 |
|---|---|---|---|
| A Length | 31' | 31' | 31' |
| B Width | 9'10" | 9'11" | 10'6" |
| C Tram Height | 52" | 49" | 46" |
| D Ground Clearance* | 6" | 6" | 6" |
| E Discharge Clearance | 28" | 28" | 28" |
| F Conveyor Width | 44" standard (optional 50" or 56") | | |
| **Production Capacities** | | | |
| Throughput (Nominal) | 400 tph | 400 tph | 400 tph |
| Hopper Capacity** (without sideboards) | 315 Cu. Ft. | 315 Cu. Ft. | 335 Cu. Ft. |
| Input Size | 2½'×3'×4' | 2½'×3'×4' | 2½'×3'×4' |
| Output Size | Nominal –6" to –8" | | |
| **Breaker** | | | |
| Drive | Mechanical | Mechanical | Mechanical |
| Drive Train Protection | Shear Sprocket/Shear Pin | | |
| Construction | Carbide tipped picks mounted on solid alloy steel shaft | | |
| Horsepower | Mech. 75 Hyd. 100 | Mech. 100 Hyd. 125 | Hyd. only up to 200 |
| Force at Pick Point | 50,000 lbs. | 68,000 lbs. | 83,000 lbs. |
| **Conveyor** | | | |
| Drive | Mechanical or Hydraulic | Mechanical or Hydraulic | Hydraulic Only |
| Construction | Solid alloy steel flights driven by high strength chain specially manufactured for feeder-breaker use. | | |
| Drive Train Protection | Mechanical Drive: Shear sprocket/Shear pin<br>Hydraulic Drive: Relief valve | | |

Dimensions shown are nominal and may be altered to meet customer's needs.
*Variable Ground Clearance option available from 0" to 18" in 8" increments.
**Hopper capacities increased with the addition of sideboards.

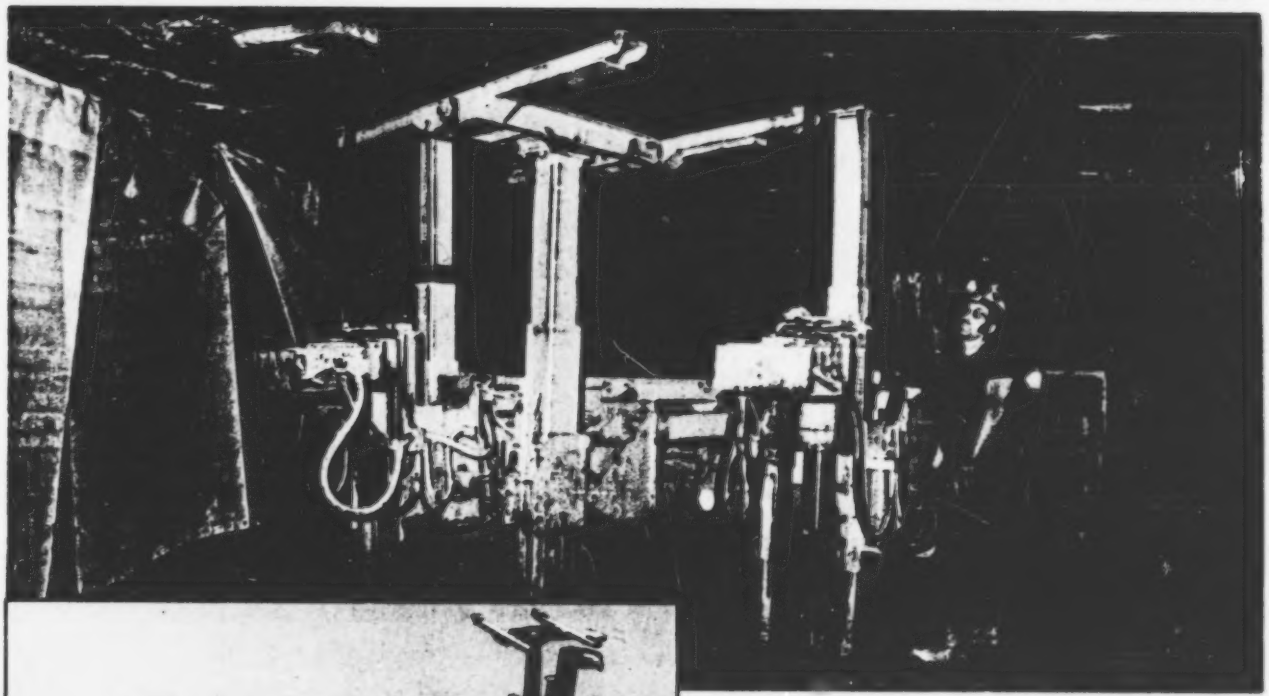
### Floor Plan and Elevation Diagram

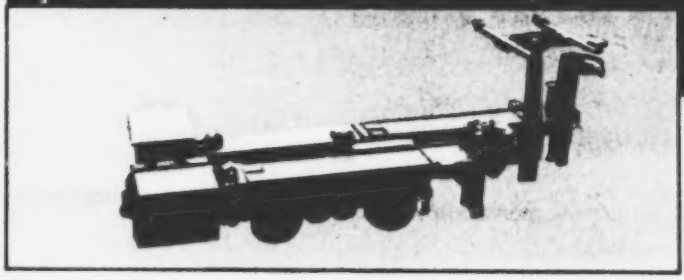



Protective guards and covers not shown to show features.
Shown with hydraulically driven conveyor. Available with mechanical drive, except Series 14-6.
*Widths shown are standard. Other widths available upon request.

# FLETCHER

## Model DDM Dual Head Roof Bolter





- Bolts twenty-foot cut with only one machine move.
- Optional TRS eliminates need for temporary posts and jacks in most applications.
- Drill masts rotate up to 45 degrees for angle drilling (optional).

With Fletcher Model DDM, you get the most flexible big-job drill for medium- and high-seam applications. Each boom telescopes and swings independently for speed and versatility; optional TRS keeps workers under supported roof at all times. Solid Fletcher design means long life and high uptime.

## Standard Equipment:

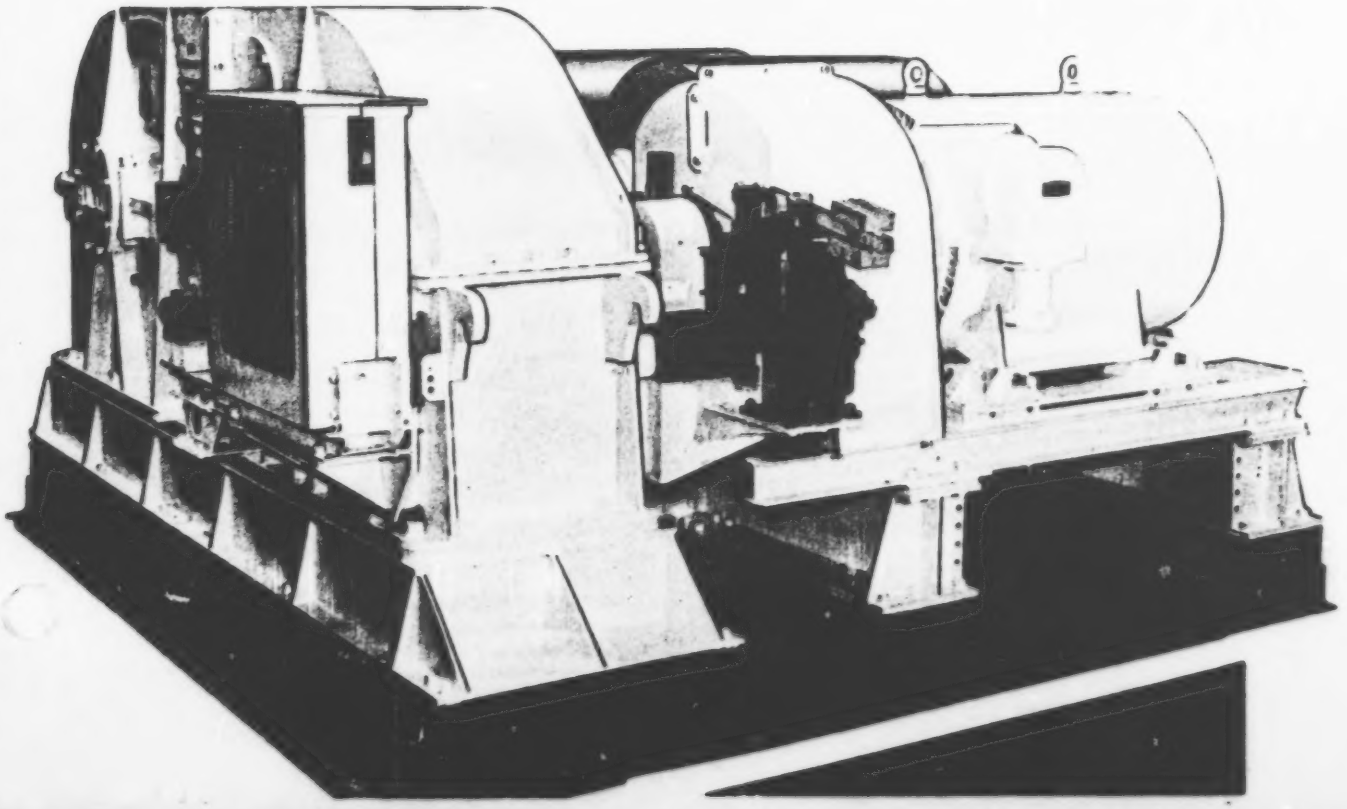
Automatic Disc Brakes
Donaldson Dust Tanks
Four-Wheel Drive
In-Line Hydraulic Filters
JIC or IPT Fittings
Separate Bolt Torque Valve
Cable Reel
Panic Bars

## Features:

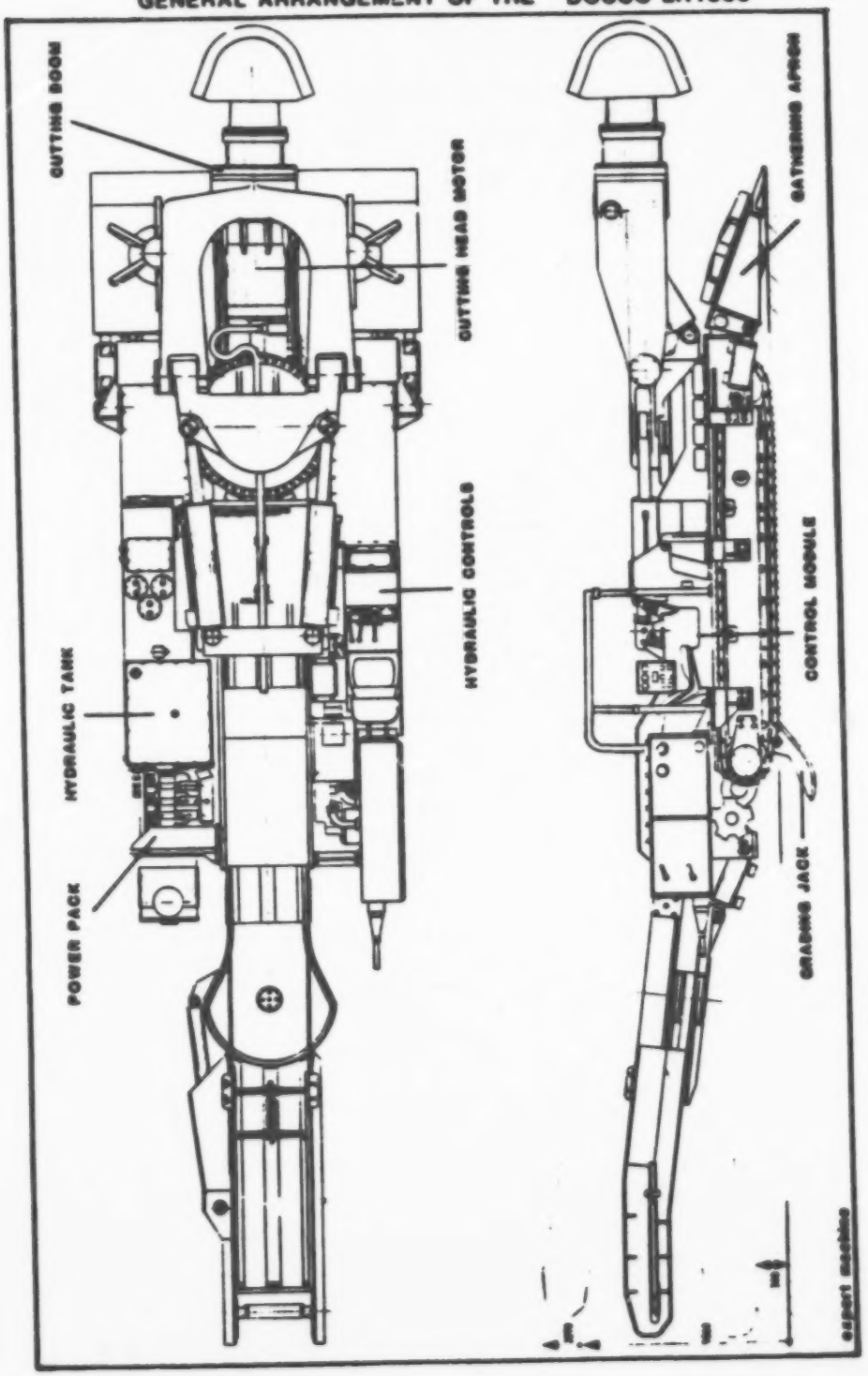
| | |
|---|---|
| Boom Swing 6'3" either side of center | |
| Telescopic Length 9'0" | 2743 mm |
| Overall Length 22'6" | 6858 mm |
| Overall Width 8'0" | 2438 mm |
| Overall Height 54" | 1372 mm |
| Clearance 11" | 279 mm |
| Weight 34,000# | 15,422 kg |

| NOMINAL BELT SPEEDS and MECHANICAL HORSEPOWER RATINGS | | | | | | | | |
|---|---|---|---|---|---|---|---|---|
| MOTOR SPEED | 970 R.P.M. | | 1160 R.P.M. | | 1450 R.P.M. | | 1740 R.P.M. | |
| GEARBOX | BELT SPEED FT/MIN M SEC | H.P. RATING | BELT SPEED FT/MIN M SEC | H.P. RATING | BELT SPEED FT/MIN M SEC | H.P. RATING | BELT SPEED FT/MIN M SEC | H.P. RATING |
| D1683 1 | 210 1.07 | 180 | 340 1.22 | 210 | 300 1.53 | 250 | 360 1.83 | 290 |
| D1683 2 | 230 1.17 | 200 | 280 1.42 | 240 | 350 1.78 | 280 | 420 2.13 | 330 |
| D1683 3 | 270 1.37 | 230 | 320 1.62 | 260 | 400 2.03 | 310 | 480 2.43 | 370 |
| D1683 4 | 300 1.53 | 250 | 360 1.83 | 290 | 450 2.28 | 350 | 540 2.74 | 410 |
| D1683 6 | 340 1.72 | 270 | 400 2.03 | 320 | 500 2.54 | 380 | 600 3.05 | 450 |
| D1683 7 | 370 1.88 | 300 | 450 2.28 | 350 | 560 2.84 | 410 | 670 3.40 | 490 |
| D1683 5 | 410 2.08 | 310 | 490 2.49 | 380 | 610 3.10 | 450 | 740 3.75 | 530 |

# The HUWOOD TB 300/500 Drive Head



## GENERAL ARRANGEMENT OF THE DOSCO LH1300



# APPENDIX 2-9

# PLANS

APPROXIMATE MINING BOUNDARY

CONTOUR REVISION IN THE DDH CBI-29 AREA

0 200 400 600 800 1000 feet
0 75 150 225 300 metres

1000 feet
300 metres

McINTYRE MINES LIMITED
COAL DIVISION

9G MINE NO.4 SEAM
SEAM COVER ISOPACH - VERTICAL THICKNESS
CONTOUR INTERVAL 50'

DATE: JUNE 16/82
NOV. 1982
PLAN No. 1

CONTOUR REVISION IN THE DDH CBI-29 AREA — 1/9/82 VGB
" " " " " " " — 15/9/82 VGB
DRAWN BY: V.G. BETTS
TRACED BY:
APPROVED BY:
DATE BY SCALE: 1" = 800'
Number

APPROXIMATE MINING BOUNDARY

CONTOUR REVISION — 31/8/82 VGB
CONTOUR REVISION — 15/9/82 VGB
DRAWN BY: V.G. BETTS
TRACED BY:
APPROVED BY:
DATE BY SCALE: 1" = 800'
Number

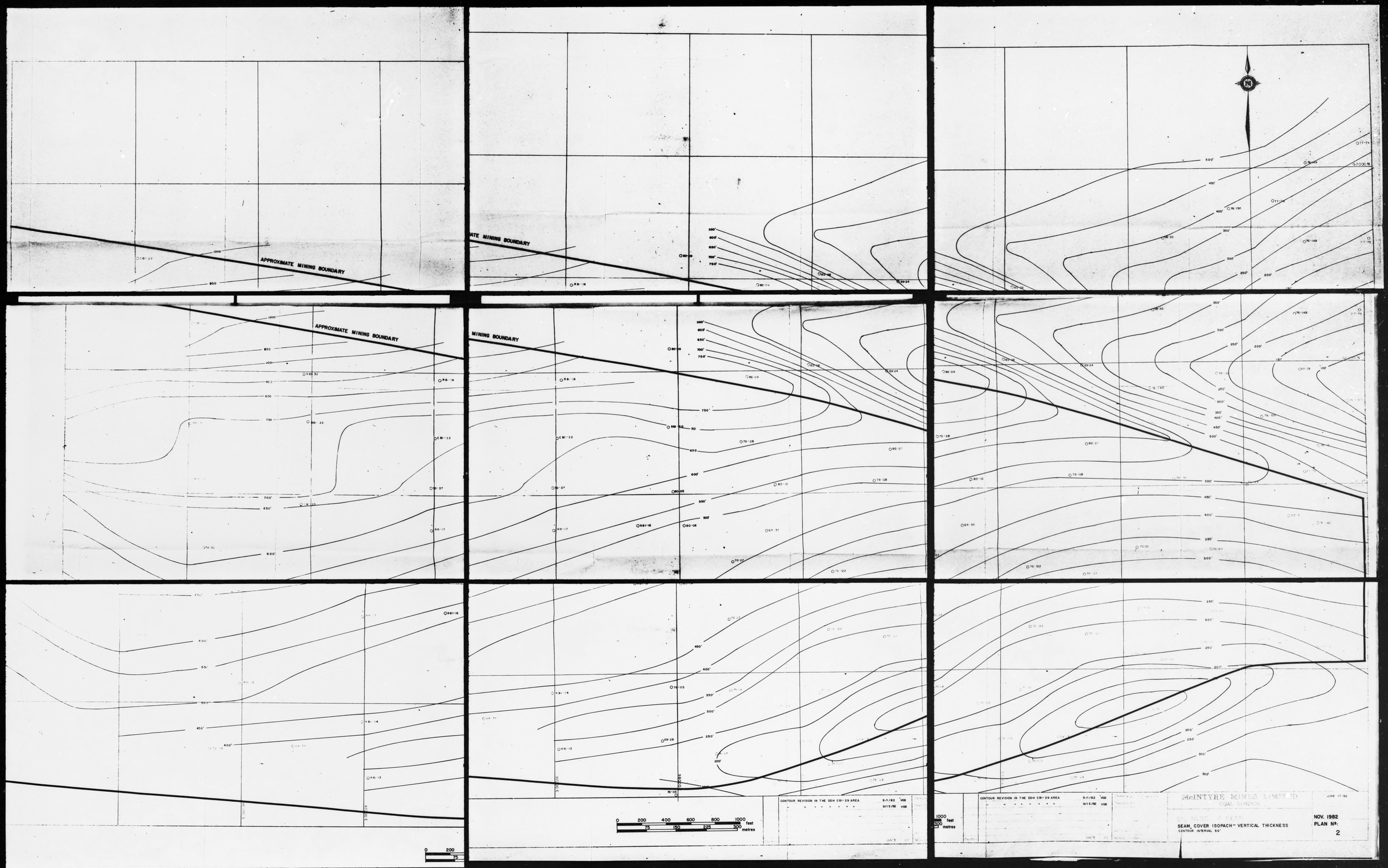
0 200 400 600 800 1000 feet
0 75 150 225 300 metres

1000 feet
300 metres

McINTYRE MINES LIMITED
COAL DIVISION

9G MINE
4 SEAM TO 10 SEAM INTERBURDEN (VERTICAL THICKNESS)
CONTOUR INTERVAL 10'

DATE: JUNE 16/82
NOV. 1982
PLAN No. 3


APPROXIMATE MINING BOUNDARY
MINING BOUNDARY
57000 N
56000 N
55000 N
54000 N
53000 N
CONTOUR REVISION IN THE DDH C81-29 AREA 9/1/82 VGB
9/15/82 VGB
0 200 400 600 800 1000 feet
75 150 225 300 metres
1000 feet
500 metres
McINTYRE MINES LIMITED
COAL DIVISION
SEAM COVER ISOPACH - VERTICAL THICKNESS
CONTOUR INTERVAL 50'
NOV. 1982
PLAN No: 2
ISOPACH REVISION IN THE DDH C81-29 AREA 30/8/82 VGB
96 MINE 10 SEAM
VERTICAL THICKNESS ISOPACH
DRAWN BY V.S. BETTS
TRACED BY
APPROVED BY
SCALE 1" = 200'
DATE FEB 24/82
NOV. 1982
PLAN No: 4

APPROXIMATE MINING BOUNDARY

MINING BOUNDARY

CBI-29 NO INTERSECTION

0 200 400 600 800 1000 feet
0 75 150 225 300 metres

| | CONTOUR REVISION IN THE DDH CBI-29 AREA | 3/9/82 | VGB | DRAWN BY: V.G. BETTS |
|---|---|---|---|---|
| | | | | TRACED BY: |
| | | | | APPROVED BY: |
| Number | | DATE | BY | SCALE: 1" = 200' |

**McINTYRE MINES LIMITED**
COAL DIVISION

9G MINE NO. 4 SEAM
VERTICAL THICKNESS ISOPACH

DATE: FEB 25/82
NO. 4 SEAM
PLAN No:

APPROXIMATE MINING BOUNDARY

BARRETT ANTICLINE

TAYLOR SYNCLINE

DRILL CAMP ANTICLINE

NOTE: DRILL HOLE LOCATIONS ARE THE INTERSECTION WITH THE TOP OF THE SEAM, NOT THE SURFACE

0 200 400 600 800 1000 feet
0 75 150 225 300 metres

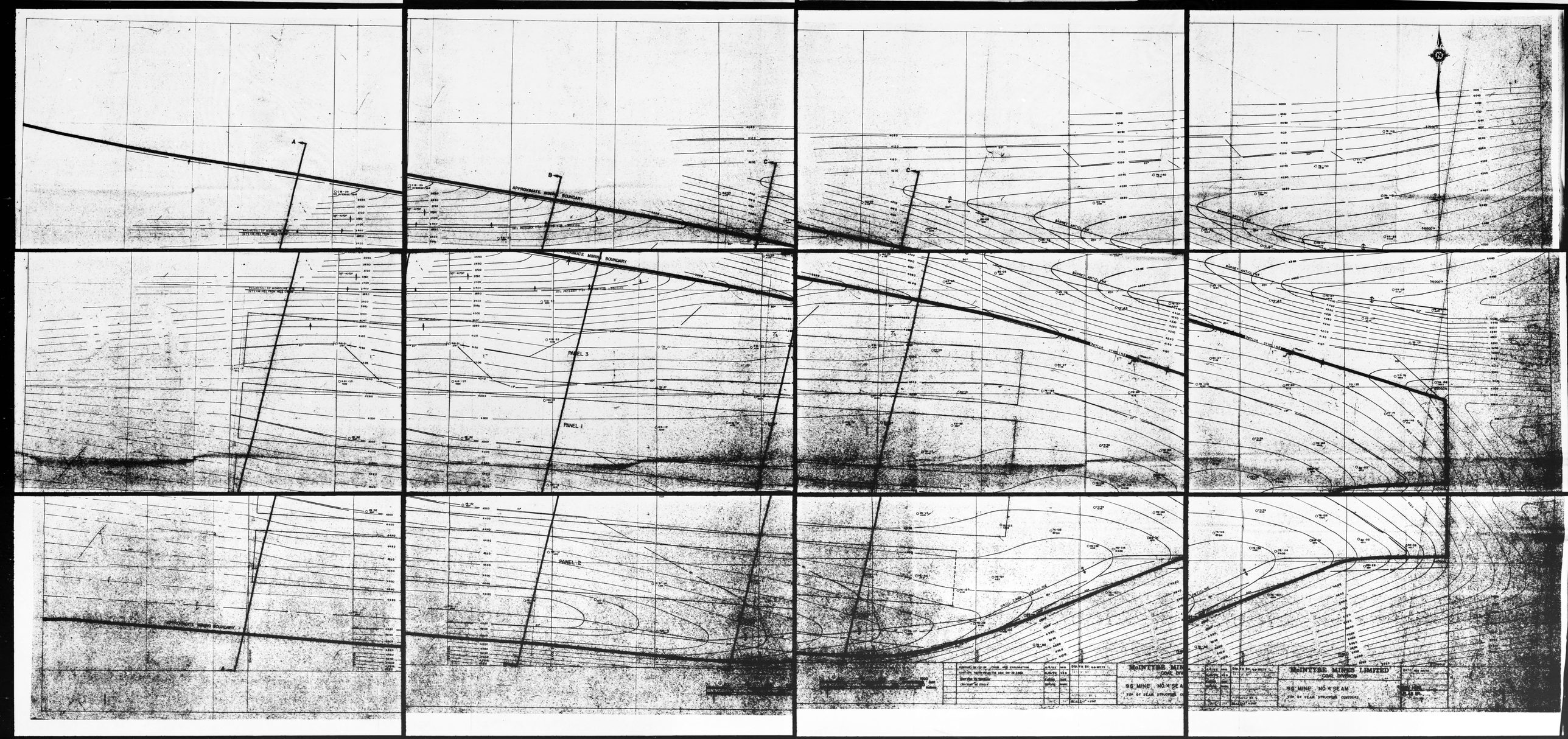
| | | | |
|---|---|---|---|
| CONTOUR REVISION FROM 1981 DRILLING | 10/3/82 | VGB | DRAWN BY V.S. BETTS |
| CONTOUR REVISION IN DDH C-81-29 AREA | 30/9/82 | VGB | TRACED BY |
| " " " " " | 15/9/82 | VGB | |
| | | | APPROVED BY |
| Number | DATE | BY | SCALE 1"= 200' |

**McINTYRE MINES LIMITED**
COAL DIVISION

# 9G MINE 10 SEAM

TOP OF SEAM STRUCTURE CONTOUR

DATE FEB 24/82
NOV. 1982
PLAN Nº: 6


A
B
C
APPROXIMATE MINING BOUNDARY
PANEL 3
PANEL 1
PANEL 2
BARRETT ANTICLINE
PATTICK SYNCLINE
McINTYRE MINES LIMITED
COAL DIVISION
96 MINE NO. 4 SEAM
TOP OF SEAM STRUCTURE CONTOURS

CRI-24
CE. 5364'
90' WEST

SECTION A–A

SECTION B–B

70-34
CE. 5362'
80' WEST

RBI-12
CE. 5255
40' EAST

SECTION C–C

76-115
C.E. 5097
30' EAST

80-08
C.E. 5100
90' EAST

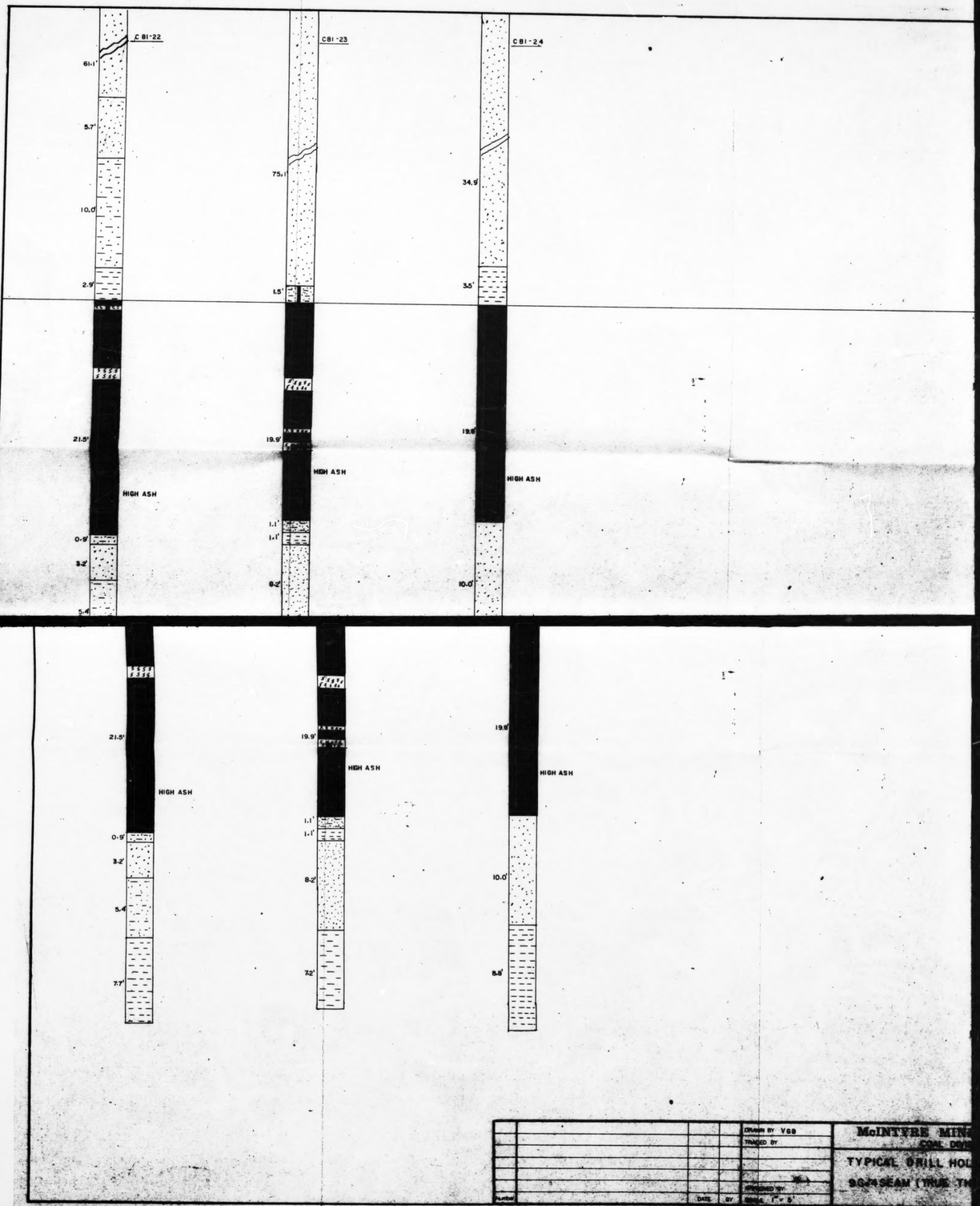
0 200 400 600 800 1000 feet
0 75 150 225 300 metres

| Number | | | | | |
|---|---|---|---|---|---|
| | SECTIONS A-A AND B-B REVISED | 3/9/82 | VGB | DRAWN BY VG BETTS | McINTYRE MINES LIMITED<br>COAL DIVISION |
| | | | | TRACED BY: | |
| | | | | | 9G MINE AREA<br>TYPICAL CROSS SECTIONS |
| | | | | APPROVED BY: | |
| | | DATE | BY | SCALE: 1" = 200' | |

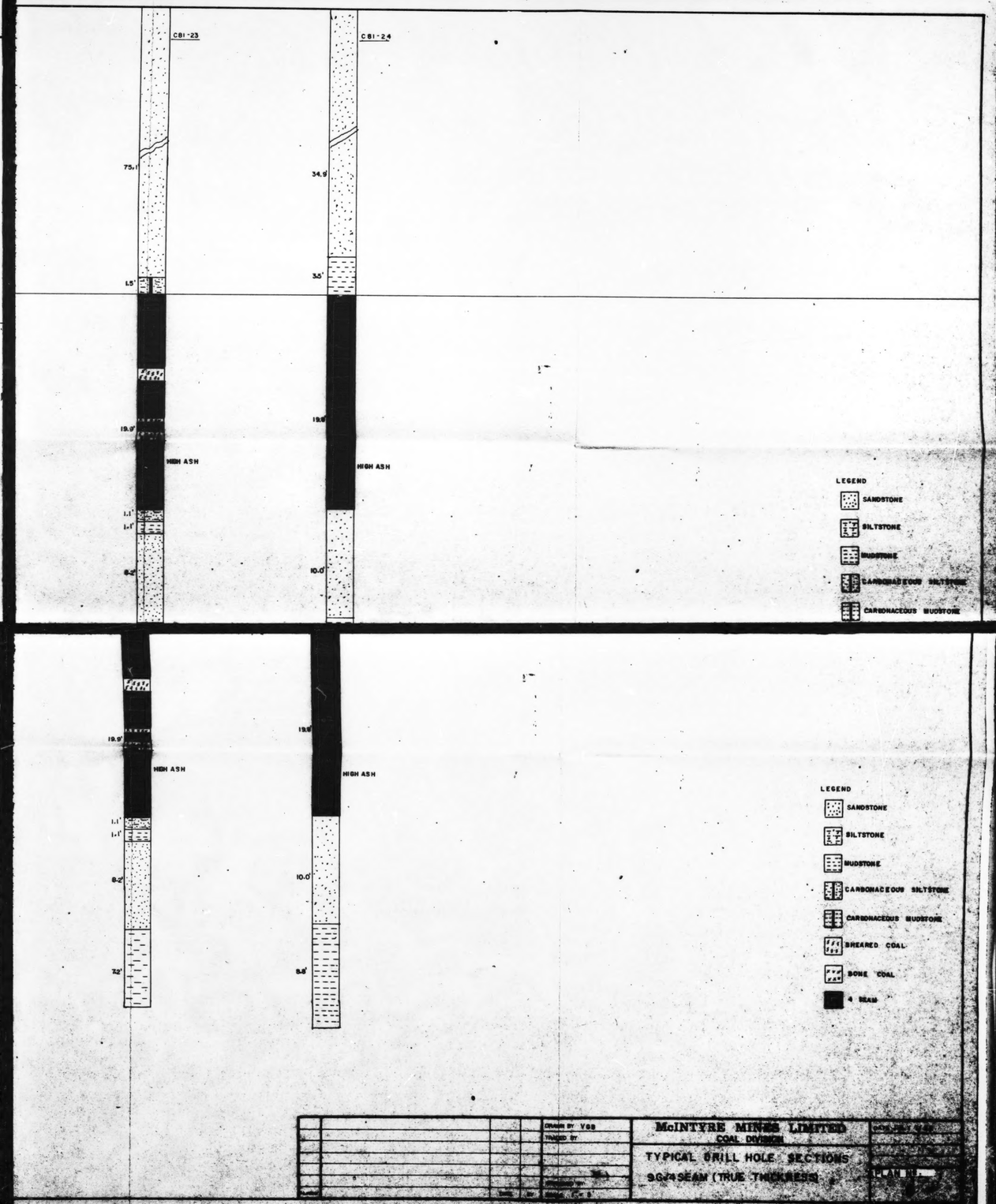
DATE: AUGUST 11/82

NOV. 1982

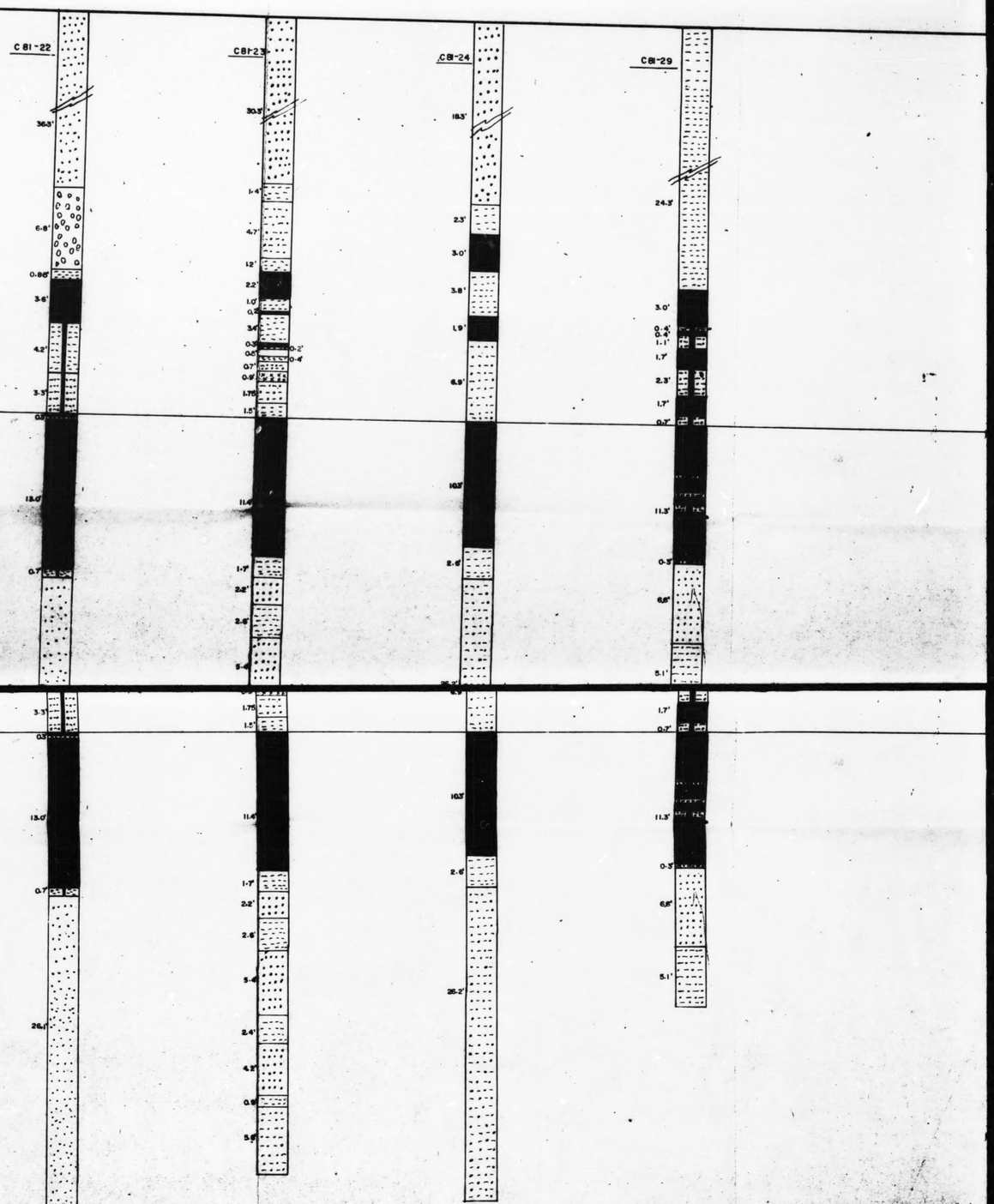
PLAN Nº: 8


C81-22
61.1'
5.7'
10.0'
2.9'
21.5'
HIGH ASH
0.9'
3.2'
5.4'
7.7'
C81-23
75.1'
1.5'
19.9'
HIGH ASH
1.1'
1.1'
8.2'
7.2'
C81-24
34.9'
3.5'
19.8'
HIGH ASH
10.0'
8.8'

DRAWN BY VGB
TRACED BY
DATE
BY
McINTYRE MINES LIMITED
COAL DIVISION
TYPICAL DRILL HOLE SECTIONS
9G/4 SEAM (TRUE THICKNESS)


C81-23
75.1'
1.5'
19.9'
HIGH ASH
1.1'
1.1'
8.2'
7.2'
C81-24
34.9'
3.5'
19.8'
HIGH ASH
10.0'
8.8'
LEGEND
SANDSTONE
SILTSTONE
MUDSTONE
CARBONACEOUS SILTSTONE
CARBONACEOUS MUDSTONE
BREARED COAL
BONE COAL
4 SEAM

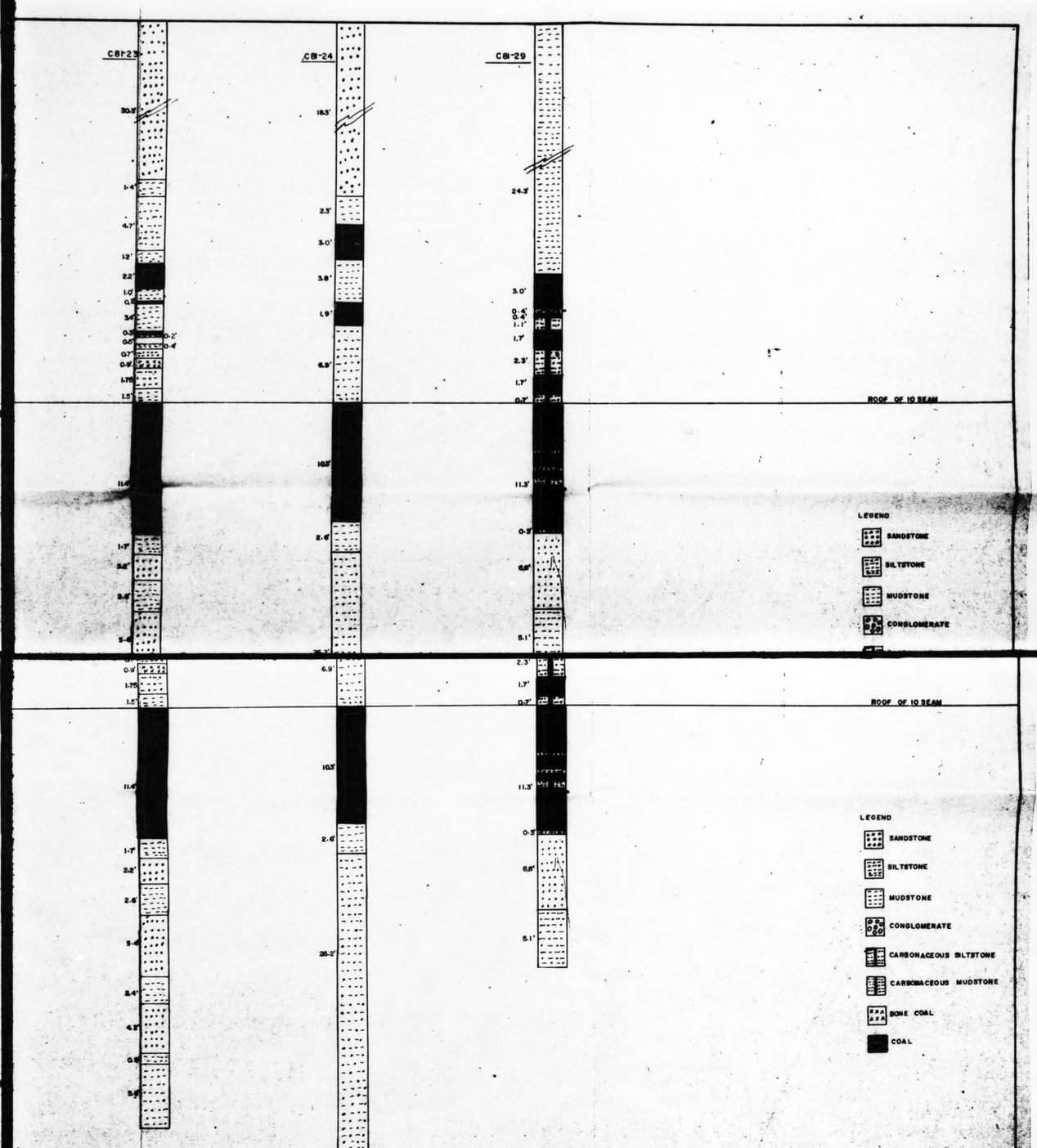
DRAWN BY VGB
TRACED BY
DATE
BY
McINTYRE MINES LIMITED
COAL DIVISION
TYPICAL DRILL HOLE SECTIONS
9G/4 SEAM (TRUE THICKNESS)




| | REVISIED HOLE C81-29 | 3/9/82 | VGB | DRAWN BY V G BETTS |
|---|---|---|---|---|
| | | | | TRACED BY |
| | | | | |
| | | | | APPROVED BY |
| Number | | DATE | BY | SCALE 1" = 5' |

McINTYRE MINES
COAL DIVISION

TYPICAL DRILL HOLE S
9G-10 SEAM (TRUE TH




| | REVISIED HOLE C81-29 | 3/9/82 | VGB | DRAWN BY V G BETTS |
|---|---|---|---|---|
| | | | | TRACED BY |
| | | | | |
| | | | | APPROVED BY |
| Number | | DATE | BY | SCALE 1" = 5' |

McINTYRE MINES LIMITED
COAL DIVISION

TYPICAL DRILL HOLE SECTIONS
9G-10 SEAM (TRUE THICKNESS)

DATE: AUGUST, 1982

PLAN No.
10

POOR

QUALITY

ORIGINAL